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THE WITWATERSRAND GOLDFIELDS

BANKET AND MINING PRACTICE

A Miner at Work in the Stopes.

THE WITWATERSRAND GOLDFIELDS

BASED ON MINE

ON THE
PARKWA

A Miner at Work in the Stopes.

THE WITWATERSRAND GOLDFIELDS

BANKET & MINING PRACTICE

WITH AN APPENDIX ON THE BANKET OF THE TARKWA GOLDFIELD, WEST AFRICA

BY

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PREFACE TO THE SECOND EDITION

THIS Second Edition has been issued in order to satisfy a very gratifying demand. My apologies are due that, owing to want of time and opportunity, no revision has been possible. My present purpose is to proceed to the Witwatersrand again, with the intention of applying myself to the work of a complete revision, and to the work of compiling another book on "Reduction Practice."

In the meantime I beg to present a short chapter on the banket of the Tarkwa Goldfield, West Africa, which embodies my own notes made during a visit to that goldfield, together with the information which others have kindly given me. In this respect I take much pleasure in acknowledging my especial indebtedness to Messrs. Stanley Clay, Leslie Simson, and W. Fischer Wilkinson.

S. J. TRUSCOTT.

LONDON, *June* 1902.

THE Third Edition is a verbatim reprint of the Second.

May 1907.

PREFACE TO THE FIRST EDITION

THE idea of writing this book grew with the desire to consolidate the notes which, during a period of about four years, I made whilst following my profession as a mining engineer on the Witwatersrand Goldfields.

As a result of the comparison of methods which has been rendered possible by the unique proximity of the mines to one another, the work on these goldfields has reached a high place in modern mining, reflecting much credit upon those engineers who have had the direction of the work in their hands.

In this work the particular occurrence of the gold-bearing ore, the banket, is described, and the mining operations which are performed in its exploitation are followed, till the ore arrives at the mill ore-bins.

I take much pleasure in expressing my great indebtedness to the mining engineers, amongst whom I would like to mention Messrs. H. R. Skinner, J. H. Johns, Strangman Hancock, F. H. P. Cresswell, S. Penlerick, T. H. Leggett, S. B. Asher, N. Wilson, and R. P. Whitelaw, who have supplied me with so much of the information here recorded. To Mr. John Hays Hammond, Consulting Engineer of the

Consolidated Goldfields of South Africa, I am especially indebted for the chapter on "The Genesis of the Witwatersrand Banket" and for kind assistance throughout; and to Professor Henry Louis, of the Durham College of Science, who has undertaken the task of revision, and has piloted the book through the press, I am grateful and indebted, more especially because the undertaking has been more than usually laborious.

I am also indebted to the *South African Mining Journal* and to the mining columns of the *Standard and Diggers' News* and the *Johannesburg Star* for many references not otherwise acknowledged, and to the Mining Companies through the Secretaries, for the copies of the Annual Reports which I received from them.

At some later date I hope to be able to cover the entire range of operations of the Witwatersrand Goldfields by adding to this work another, on Metallurgical Practice, which is taken to start with milling.

S. J. TRUSCOTT.

LONDON, 29*th January* 1898.

CONTENTS

CHAPTER I

INTRODUCTION

CHAPTER II

GEOLOGY

	PAGE
WITWATERSRAND SYNCLINE	5
Superficial Extension	5
Central District	6
Van Ryn-Chimes District	6
Nigel-Heidelberg ,,	6
Venterskroon ,,	8
Orange River Colony	8
Klerksdorp District	8
Krugersdorp ,,	8
Summary	9
Inclination	11
SOUTHERN HEIDELBERG SYNCLINE	17
GEOLOGICAL STRUCTURE—	
The Granite	18
Quartzite-Shale Group	20
The Witwatersrand Beds	23
Amygdaloidal Diabase	25
The Black Reef Formation	29
The Dolomite Formation	32
The Gatsrand Formation	33
Coal Measures	34
Summary	34
Age of the Witwatersrand Beds	36

CHAPTER III

Banket Beds and Banket

PAGE

Banket in the Quartzite-Shale Group 41
Banket in the Witwatersrand Beds 43
Central District 43
Main Reef Series 44
Livingstone Series 45
Bird Reef Series 46
Kimberley Series 47
Elsburg Series 49
Van Ryn-Chimes District 50
Main Reef Series 50
Chimes Series 50
Bird Reef Series 52
Kimberley Series 52
Elsburg Series 52
Bird Reef Series 53
Chimes Series 53
Main Reef Series 53
Nigel and Heidelberg Districts 53
Nigel Reef 54
Battery Reef 54
Joel Reef 55
Central Reef 55
Elsburg Reef 56
Venterskroon District 56
Klerksdorp District 57
Large-Pebbled Reefs 58
Small-Pebbled Reefs 60
Krugersdorp District 60
Botha's Series 61
Africander Series 62
Monarch Series 62
Battery Reef Series 62
Elsburg Series 63
Southern Heidelberg Syncline 63
Nigel Reef 63
Battery Reef 64
Joel Reef 64
The Central Reef 64

PAGE

THE BLACK REEF FORMATION 64
Klerksdorp District 64
Krugersdorp District 66
Black Reef District 66

CHAPTER IV

MAIN REEF SERIES

MAIN REEF SERIES 68
BASTARD REEFS 92

CHAPTER V

DISLOCATION

TRANSVERSE DISLOCATIONS 96
LONGITUDINAL DISLOCATIONS 100
Ferreira Crown Deep Dyke 101
Grahamstown Dyke 101
South Rand Dyke 102
George Goch Dyke 102
Metropolitan Fault 103
Simmer Dyke 103
Knights Longitudinal Dyke 104
East Rand Longitudinal Dyke 104
Balmoral Longitudinal Dyke 105
Village Main Reef Fault 105
BEDDED DISLOCATIONS 108

CHAPTER VI

"THE GENESIS OF THE WITWATERSRAND BANKET"

By John Hays Hammond 123

CHAPTER VII

PROSPECTING

TOPOGRAPHY 129
GENERAL CONSIDERATIONS 130

PAGE
TRENCHING 130
DIAMOND DRILLING 131
Construction of Diamond Drill 132
Details of Bore-holes 138
Lancaster Bore-hole 138
Crown Deep „ 138
Bezuidenville „ 139
Nourse Deep „ 139
Rand Victoria „ 139
Rose Deep „ 139
Simmer East „ 139
Chimes Mines „ 139
Assay of Reef Cores 139
Depth and Dip from Reef Cores 141
The Surveying of Bore-holes 143
Records of Bore-hole Results 144
Boring Returns 146
PROSPECTING BY CROSS-CUTS 148
PROSPECTING SHAFTS 148

CHAPTER VIII

SHAFTS, SINKING, AND TIMBERING

LOCATION OF SHAFTS 151
SIZES OF VERTICAL SHAFTS 152
SIZES OF INCLINE SHAFTS 155
SINKING 155
SINKING AN INCLINE SHAFT 164
TIMBERING THE SHAFT 165
Wall Plates and End Plates 166
Dividers 168
Guides 169
Studdles 171
Lagging 173
Fixing and supporting the Timbers 173
Hanging Bolts 175
Bearers 175
Incline Shaft Timbering 178
Timbering of the Angle Connection 184
CIRCULAR SHAFTS 186

PAGE
SUMPS, LODGES, ETC. 188
RATES OF SINKING AND COSTS OF SHAFTS 189
COMPARISON OF MACHINE AND HAND DRILLING IN SINKING . 193

CHAPTER IX

UNDERGROUND STATIONS AND ORE BINS

STATIONS 197
ORE BINS 203
Capacity of Bins 211
Ore-Bin Doors and Shoots 213

CHAPTER X

SHAFT TOPS AND HEADGEARS

HEADGEARS 218
Tipping Arrangements 229
Construction of Headgears 237

CHAPTER XI

WINDING APPLIANCES

SKIPS, CAGES, AND BUCKETS 244
Buckets 244
Skips 246
Cages 257
DOGS AND GATES 258
RAILS, GUIDES, PULLEYS, ETC. 259
ROPES 260
WINDING ENGINES 261
CONSIDERATIONS IN WINDING 266
COSTS OF HOISTING 268

CHAPTER XII

PUMPS AND PUMPING

CORNISH PUMPS 275
STEAM PUMPS 283

PAGE
FORCE PUMPS 284
COSTS OF PUMPING 284

CHAPTER XIII

DEVELOPMENT

DRIVES 286
DRIVING 288
WINZES, SINKING AND RAISING 292
CROSS-CUTS AND CROSS-CUTTING 294
COSTS OF DEVELOPMENT 296

CHAPTER XIV

SAMPLING

UNBROKEN ORE 305
System of Sampling 305
Practical Sampling 306
Sample Grinding 310
Panning 311
Mine Sampling Records 312
Delineation of Dykes and Faults 317
Averaging Values 318
Samples taken regularly 320
Samples taken irregularly 321
Determination of the Value of Areas 325
Comparison of Results 328
Trial Crushings 329
SAMPLING OF BROKEN ORE 330
DETERMINATION OF ORE VALUE BY REDUCTION RESULTS . . 334

CHAPTER XV

STOPES AND STOPING

STOPES 335
Underhand Stoping 336
Overhand Stoping 339
Combined Stoping 342

STOPES— PAGE
Breast or Side Stopes 342
Longwall Stopes 343
Method of working Reefs which are close together . . . 343
Width of Stopes 345
SUPPORTING THE ROOF 346
Pillars 346
Level Pillars 346
Stope Pillars 347
Shaft Pillars 348
Boundary Pillars 349
Stulls 349
Timbering 354
STOPE BOXES 354
REMOVING THE BROKEN ORE 356
BREAKING THE ORE IN THE STOPE FACE 357
EXPLOSIVES 361
LABOUR IN THE STOPES 362
STOPE PLANS AND ORE ACCOUNT 364

CHAPTER XVI

AIR-COMPRESSORS, ROCK-DRILLS, ETC.

AIR-COMPRESSORS 366
RECEIVERS 373
AIR MAINS AND PIPES 373
MACHINE DRILLS 375
The Slugger 377
Ingersoll Eclipse 377
Climax 377
Ingersoll Sergeant 377
DRILL STEEL AND BITS 381
RATES AND COSTS 382

CHAPTER XVII

VENTILATION AND ILLUMINATION

VENTILATION 387
ILLUMINATION 391

CHAPTER XVIII

TRANSPORT AND TRAMMING

PAGE
RAILS 394
TRUCKS 395
MECHANICAL HAULAGE, ETC. 399

CHAPTER XIX

ORE-DRESSING

SCREENING 407
SORTING 407
Underground Sorting 412
Principles of Sorting 413
Methods of Sorting 413
Sorting on Floors 414
Sorting on Revolving Tables 417
Sorting on Moving Belts 420
Sorting on a Shaking Table 422
Comparison of Methods 423
CRUSHING 425
Gates Crusher 425
Blake Crusher 426

CHAPTER XX

ADMINISTRATION, LABOUR AND MATERIAL

ADMINISTRATION 429
Crown Reef Accounts 432
Witwatersrand ,, ... 437
New Comet ,, ... 439
Geldenhuis Estate Accounts 440
GROUND TENURE 447
LABOUR 448
White Labour 449
Native Labour 453
STORES AND MATERIAL 457

CHAPTER XXI

PAGE

DEEP-LEVEL MINING 459

APPENDIX I

Mining Regulations, Section XXI. 470
" " " XXII. 471
" " " XXIII. 473
Regulations for the Examination of Mine Managers . . . 474
Regulations for the Examination of Mine Overseers . . . 476
Regulations for the Examination of Engine-Drivers . . . 476
Advantages of Sorting 478
Steel Ropes 480
Working Costs at the Ferreira 482
Ore Tonnage 484
Reduction of Working Costs 485

APPENDIX II

THE BANKET OF THE TARKWA GOLDFIELD, WEST AFRICA

Position and Extent 487
Geological Structure and Formation 490
Displacement, Dykes, and Faults 495
Details of the Banket 495
The Tarkwa Reef 496
The Formation at Sekondi 505
Geological Age 505
Origin of the Banket 506
Genesis of the Gold 506

INDEX 509

ILLUSTRATIONS

FIG.		PAGE
	A miner at work in the Stopes	*Frontispiece*
	Rough map of the formations which either conform to or are found in the Witwatersrand and Southern Heidelberg Synclines	*To face* 8
1.	Section across formation at Johannesburg	„ 44
2.	Section across the formation from the Vogelstruis Estate to the Great Britain	*To face* 44
3.	Section of the Bird Reef Series as projected at a dip of 32°; from the results of a bore-hole put down south of the Wemmer	46
4.	Section of the Bird Reef Series as obtained on surface near the Crown Reef Battery	47
5.	Section through the Kimberley Series of the Great Britain Mine	48
6.	Section across the formation principally projected from the Chimes Mines bore-hole	*To face* 50
7.	Section across the formation at the Nigel reproduced from a vertical section	*To face* 50
8.	Section of the Chimes series in the Chimes mine	51
9.	Section across the Van Ryn West	52
10.	Section across the outcrop of the Witwatersrand Beds in the Venterskroon district	56
11.	Section across the formation about one mile east of Krugersdorp from the Botha's Series to the Battery Series	*To face* 62
12.	Section across the Southern Heidelberg Syncline	„ 62
13.	Section of the Main Reef Series in No. 2 shaft, Robinson Deep	73
14.	Section showing Main Reef Leader with the strong parting in its footwall and the Main Reef, in a stope on the 4th level George Goch No. 3 mine	75
15.	Section of Reef Series obtained from a bore-hole on the Nourse Deep Property	76
16.	Section of Main Reef Series in a cross-cut south from the New Heriot shaft	76
17.	Section of the Main Reef Series in the Percy section of the Treasury	77
18.	Section along the cross-cut south, on the 2nd level from the old vertical shaft of the Geldenhuis Estate	79
19.	Section of reefs in the western portion of the Geldenhuis Estate	79
„	„ „ eastern „ „ „	79

FIG. PAGE

20. Section across the South Series, 3rd level, central section Witwatersrand mine 80
21. Section of North Series of reefs, New Comet mine, 4th level cross-cut 82
22. Section of portion of the South Reef Series exposed 4th-level Phœnix shaft, New Comet 83
23. Section 550 feet west of No. 4 shaft, Van Ryn Estate . . 87
24. Section at No. 4 shaft, Van Ryn Estate 87
25. Section at Main Shaft, to the east of No. 4 shaft, Van Ryn Estate . 88
26. Section of the Main Reef Series obtained in the West Roodepoort Deep 89
27. Section obtained from north cross-cut, 560 feet level, No. 1 shaft, Botha's Series, Lancaster mine 91
28. Section of Main Reef, Botha's Series, obtained in the western bore-hole, Lancaster mine 91
29. Diagram showing the extension of the three principal reefs of the Main Reef Series . . . . . *To face* 92
30. Normal and reverse faults 95
31. Diagram of fault 95
32. Vertical section, chiefly from surface data, across the formation from the Main Reef Series on the Crown Reef, to the Kimberley Series on Booysens Estate along a line indicated on the Dyke map . . . . . *To face* 100
Map of the Central Rand showing the main longitudinal dykes „ 100
33. Section through the Main Shaft of the Spes Bona, showing the occurrence of a bedded dyke 109
34. Occurrence of an interbedded dyke between the 5th and 6th levels of the Worcester 111
35. Occurrence of a gold-bearing bedded dyke in the Ferreira . . 112
36. Plan of the outcrop portion of the Wemmer, showing the intersection of dykes, faults and reefs with the plane of the surface . . 114
37. New Blue Sky, East Section, cross section showing reefs and dyke matter 114
38. Sections through the two shafts of the Ferreira mine showing the upper and lower intersections of the Ferreira Crown Deep dyke with the Salisbury dyke 116
39. Section of the formation afforded by south cross-cut, 5th level, Langlaagte United 117
40. Sullivan drill, size "B" 134
41. The American Rock Drill Company's No. 3 drill . . . 135
42. The M. C. Bullock drill "Dauntless" 136
43. Short's hand-power drill 137
44. Diagram illustrating the effect of deviation from the perpendicular upon the angle which the bedding planes make with the direction of the bore-hole 142
45. Graphic record of a bore-hole 145
46. Diagram showing the arrangement of the holes in the bottom of the Vogelstruis Deep Shafts working with machine drills . . 156

FIG. PAGE
47. Diagram showing the arrangement of the holes in the bottom of the Durban-Roodepoort Deep Shafts 158
48. Diagrammatic section of the bottom of a large shaft which is being sunk by hand labour 161
49. Diagrammatic plan and section showing the method of placing the holes in the Robinson Deep 163
50. Diagram showing the position of the cut holes in sinking the Ferreira Deep shafts 164
51. Diagram showing the arrangement of holes in the bottom of an incline shaft when drilling is done by hand labour . . 165
52. Plan of timber set, framing the shaft excavation . . . 166
53. Isometric projection of the joint between end and wall plates . 167
54. 55. } Sections through divider joints 169
56. Different shaped tenons on the dividers 170
57. Showing the attachment of guide to divider with and without a 171
distance piece 171
58. Diagram showing the studdle not flush with the outside of the set . 172
59. Diagram of headgear foundations of the five compartment shaft of the Ferreira Deep, Limited 173
60. Diagram showing the method of fixing hanging bolts between two sets 174
61. Iron bearer for wall plates used at the Vogelstruis Deep . . 177
62. Timbering of the Durban-Roodepoort Main Incline shaft . . 179
63. Timbering of City and Suburban Main Incline shaft . . 181
64. Clap-me-down joint between cap and sole pieces . . . 182
65. Shaft timbering at the angle connection between a vertical and an incline shaft, for continuous hoisting 185
66. Plan of the mouth of the Langlaagte Royal Circular shaft . . 186
67. Cubic parabolic curve proposed by R. M. Catlin for direct hauling from incline to vertical shaft ways, Hammond shaft, Simmer and Jack East, Limited 187
68. Plan and Elevation of ore bin for vertical shaft . . . 199
69. Stations, ore bin and pass in the West Incline shaft of the Robinson 200
70. Plan and section of an ore bin over an incline shaft . . . 201
71. Diagrammatic section of a station on the incline shaft of the Jubilee . 202
72. Diagrammatic section of a station on the incline shaft of the Worcester 202
73. Diagram of a station in No. 1 shaft, Robinson Deep . . 203
74. Section showing the main ore bins at the junction of the vertical portion of No. 2 shaft, Robinson Deep, with the incline portion, along which separate hoisting has been adopted . . . 204
75. Diagrammatic section of a bin over an incline shaft dipping about 60° 205
76. Plan and section of a station where the incline shaft is close to the reef 206
77. Diagrammatic section of an ore bin made over an incline shaft by cutting away the angle 207
78. Arrangement of an ore bin over an incline shaft . . . 208

FIG. PAGE
79. Ore bin and loading station for an incline shaft . . . 209
80. Section through the lower levels of the City and Suburban Main Incline shaft 210
81. Projection on to a longitudinal plane of the ore passes and ore bins of the Robinson mine 211
82. Diagram illustrating the connection between the ore bin and main drives when the shaft is on the reef 212
83. Bin or hopper door 213
84. Arrangement for loading a skip in a vertical shaft . . . 214
85. Underground bin doors at the City and Suburban mine . . 215
86. A complete loading station, showing a wheeled and a framed skip 217
87. Diagrammatic representation of the method of receiving the ore at surface, as formerly used at the Nigel mine . . . 218
General view of the surface works of the Witwatersrand mine, showing headgear with sorting and crushing house attached, mill and cyanide works . . . . . *To face* 218
88. Inclined headgear, Driefontein Consolidated . . . 221
89. Two elevations of the style of headgear which has been erected on the Rand Mines properties 222
90. Early stage in the erection of one of the Rand Mines headgears, Ferreira Deep . . . . . *To face* 222
91. Rand Mines headgear arranged with ore and waste bins and with tip for incline skips 223
92. Two elevations of a headgear erected on the Knights Central property 224
93. Steel headgear in course of erection on No. 2 shaft, Robinson Deep, over the temporary wooden headgear . . . *To face* 224
94. Ferreira headgear 225
95. Two elevations of the City and Suburban Incline headgear . . 226
96. View of the headgear which has been erected on the Knights Central Mine, two drawings of which are given in Fig. 92 . *To face* 226
97. Diagrammatic plan and elevation of the Chimes West Incline headgear 228
98. Tip on an incline headgear (Kleinfontein) 229
99. Arrangement on the headgear of a vertical shaft for the tipping of a skip carried on wheels 230
100. Tipping arrangement on a vertical headgear for an ordinary incline skip 231
101. Diagram showing the method of tipping a skip which, in a vertical shaft, is carried on a framework 232
102. Arrangement for tipping the skip on the Ferreira headgears . 232
103. Tumbler used for tipping fixed trucks 234
104. Method of temporarily breaking an incline track to provide a tip . 235
105 Method of removing the guides so that a skip carried on a frame can be taken off the winding track 236
106. Support of pulleys on the Rietfontein A headgear . . . 239
107. Headgear top, Langlaagte Royal 240

FIG. PAGE
108. Ore bins at the Chimes West headgear 241
109. Arrangement for filling trucks from an overhanging bin-face . 242
110. Sliding door for bin 243
111. Arrangement for filling trucks from a vertical face of a bin . . 243
112. Tipping bucket 244
113. Frame for guiding sinking bucket 245
114. Swivel with ball bearings to take the torsion off a rope . . 246
115. Detachable hook for a bucket 246
116. Skip for vertical shafts carried on a frame 247
117. Wheeled skip for incline shafts 249
118. Shackle, thimble and clip for the attachment of the wire rope to the draw-bar 252
119. Pitchford's self-dumping skips 253
120. Pitchford's self-dumping skip, arranged to tip backwards, and fitted with safety appliances against over-winding and rope-breaking . 254
121. Method of tipping Pitchford's skips 255
122. Cage to carry one truck in a vertical shaft 256
123. Patent safety detaching hooks for over-winding . . . 257
124. "Dogs" for supporting cages at landing stations . . . 258
125. Bottom lift pump for an incline shaft 276
126. Plan of pumping station for an incline shaft . . . 276
127. Pumping station in an incline shaft 277
128. Plan of the pump chamber of an incline shaft projected on to a plane at right angles to the plane of the shaft 278
129. Balance bob and sweep rods at the mouth of an incline shaft . 280
130. Pump spears in incline shaft 281
131. Diagram illustrating a method of setting off the plunger pole from the rods 282
132. Face of a drive drilled over for a side cut 288
133. Face of a drive drilled over where the rock is of average hardness, centre cut 289
134. Another method of drilling over the face with rock of ordinary character 289
135. Arrangement of holes in the face of a drive for easy ground . 290
136. Another arrangement of holes in the face of a drive in easy ground. 290
137. Arrangement of holes in the face of a drive where the ground is tight 291
138. Usual system of drilling over face of winze. 293
139. Face of a cross-cut drilled over in easy ground 294
140. Face of a cross-cut drilled over to take the top portion first and to leave the floor to be lifted separately afterwards . . . 295
141. Sketch showing the usual position of the reef in a drive . . 307
142. Sketches showing when samples from stopes and stope drives may be taken for computing the value of ore reserves . . . 307
143. Sample grinder 311
144. Section showing the different layers of the banket worked in the Geldenhuis Estate, and illustrating the method of averaging these different layers 315

FIG. PAGE
145. Plan of portion of a level showing the method of placing the values on the assay plan of the Geldenhuis Estate . . . 317
146. Diagrammatic plan showing the method of indicating dykes and faults in plans 318
147. Diagram illustrating the graphic method of averaging irregularly taken samples 323
148. Diagram illustrating a method of obtaining a value over an area on the reef plane 326
149. Diagrammatic longitudinal section of a mine, showing methods of stoping and the plan of development 336
150. Diagram showing the Crown Reef underhand stopes . . 337
151. Diagram showing the method of opening out an underhand stope . 338
152. A and B are overhand stopes 339
153. A B is a piece of ground which would be worked by overhand stoping, and C D by underhand stoping without connection by a winze 340
154. Diagram showing combined stopes 342
155. Section of the three reefs in some portions of the western section of New Primrose where, for a limited extent, there is a stope on each reef 343
156. Diagram showing the method of working the two stopes in the New Kleinfontein 344
157. Diagram illustrating the procedure in picking over a stull . . 351
158. Diagram illustrating the formation of a stull . . . 352
159. Horizontal section across a stope where the reef is dipping at a low angle, showing the formation of a stull by walling, and the relation of the stull to the working face 353
160. Stulling along a travelling way 353
161. Walling of waste to protect a level on a flat reef after the ground has been worked out on either side 353
162. Stope box for a reef dipping at a fair angle, with drawing of box-door 355
163. Link, lever, and pin for opening and shutting a box-door . . 355
164. Doors for stope boxes 356
165. Stope box for a reef dipping at a low angle . . . 356
166. Diagram showing the direction of the holes and the formation of benches in underhand and overhand stoping . . . 358
167. Diagram showing the method of blasting down the reef in the face of a large stope 359
168. Diagram showing method of keeping up a stope plan . . 364
169. Duplex compressor, single stage (Ingersoll Sergeant) . . 367
170. Cross compound compressor, double stage, Fraser and Chalmers (Riedler system) 367
171. Air cylinder with poppet valves, Rand Drill Company . . 370
172. Riedler air valves 371
173. Ingersoll Sergeant piston inlet air cylinder 372
174. Stewart's patent flange joint 374
175. Screw coupling joint 375

FIG. PAGE
176. Ingersoll Eclipse drill 376
177. Climax air valve drill 376
178. Ingersoll Sergeant drill 376
179. Climax tappet drill 376
180. Drill mountings 379
181. Miners' iron candle-holder 392
182. Kaffirs' wire candle-holder 392
183. Rails and sleepers of an ordinary level track . . . 395
184. Double-side "V"-tip truck (Howard, Bedford, England) . . 395
185. Double-side "round"-tip truck (Howard, Bedford, England) . 396
186. Ordinary form of axle-box 397
187. Tension carriage and weight box 398
188. Jockey fixed on to the end of a truck for mechanical wire-rope haulage 399
189. Elevation of headgear, showing sorting and crushing arrangements, etc. 405
190. Elevation of crusher station 406
191. Section of grizzly 414
192. Diagrammatic section through the Ferreira sorting floor . . 415
193. Diagrammatic plan of the Crown Reef revolving sorting table . 417
194. 24-foot circular sorting table at the Consolidated Main Reef Mines and Estate, Limited 421
195. Shaking tray or table 423
196. Gates crusher 425
197. Blake crusher 427
198. Kaffir compound of the May Consolidated Gold Mining Co. . *To face* 456
Sketch map indicating the position and extent of the Tarkwa gold-field, West Africa 489
199. Section across the formation at the Adjah Bippo mine, showing the way in which the beds may occur 491
200. Sections of the Tarkwa Reef in the Adjah Bippo mine and in the bore-holes to the dip of that mine 497
201. Section of the Tarkwa Reef at the Cinnamon Bippo mine . . 501
202. Section of the Tarkwa Reef cut in No. 2 bore-hole, Abbontiakoon . 502
203. Section of the Tarkwa Reef in the Taquah mine; second level, 50 feet south-west of No. 1 shaft 502
204. Section of the Tarkwa Reef in the Taquah mine; second level, 100 feet north-east of No. 2 shaft 503
205. Section of the Tarkwa Reef cut in No. 1 bore-hole, Effuenta . 504
Map showing the principal properties along the line of the Tarkwa Reef 507

TABLES

Analysis of Working Expenditure, Crown Reef Gold Mining Company, for the year ending 31st March 1897 . . . *To face* 432
Specimen Blank Table—Monthly Analysis of Gold Production in the South African Republic, January 1898 . . . *To face* 444
The Witwatersrand Goldfields—Output for 1897 . . *To face* 446

CHAPTER I

INTRODUCTION

THE term Witwatersrand Goldfields is taken to include all the goldfields, amongst others the Witwatersrand, Klerksdorp, Krugersdorp, Venterskroon, and Heidelberg Goldfields, which were proclaimed by the late South African Republic to cover all the occurrences of auriferous banket on which prospecting work or exploitation had been carried on. These occurrences are arranged around the borders of two synclines which exist in the southern portion of the Transvaal Colony.

The most important developments are, however, confined to the district between Randfontein, on the west of Johannesburg, and Modderfontein on the east, a distance of about 27 miles, the greater portion being on the southern slope of the Witwatersrand range of hills. It is from the mines situated between these two points that almost the whole of the gold output from the banket is being obtained. The only notable occurrences outside of these limits are the Nigel mine in the Nigel district, the Heidelberg-Roodepoort mine and the Molyneux mine in the Heidelberg district, and the various mines of the Klerksdorp district.

The unit of area used in these goldfields is the "claim," which is defined in the Gold Law of the late South African Republic in the following terms: "A quartz reef claim shall be 150 feet in length, *i.e.* along the direction of the reef, and

400 feet in breadth." The boundaries of this rectangular area extend vertically downwards to an indefinite depth to include the mineral beneath. The feet here referred to are Cape feet, and as 1 Cape foot is equivalent to 1.033 English feet, a claim in English measurements is 154.95 feet in length and 413.2 feet in breadth. The following tables show some further relations between the two systems of measurement :—

CAPE LINEAL MEASURE

12 inches	equal . . .	1 foot
12 feet	„ . . .	1 rood

CAPE SQUARE MEASURE

144 square inches	equal . .	1 square foot
144 square feet	„ . .	1 square rood
600 square roods	„ . .	1 morgen
1 morgen	„ . .	2.116 English statute acres

The most important relations of the claim to the English and Cape square measures are :—

1 claim	equals . .	64025.34 English square feet
„	„ . .	60000.00 Cape „
„	„ . .	0.6944 morgen
„	„ . .	1.46981 English acres
1 morgen	„ . .	1.43999 claims

Those mines, within the areas of which the reefs that are being worked come to the surface, are known as "outcrop mines," and in relation to these, those which are situated vertically over the immediate extension of the reefs dipping from the outcrop mines, are known as the first row of "deep-level mines." The average distance of the northern boundary of these "deep-levels" from the outcrop, along the Witwatersrand, is about 1350 feet.

Farther to the south lies the second row of deep-level mines; these are situated over the reef as it extends downwards from the first row of deep-levels; the average distance of the northern

boundary of these mines from the outcrop is about 4000 feet. For example, in the section, Fig. 32, the Crown Reef is the outcrop mine, the Crown Deep is the first deep-level mine, and the South Rand is the second deep-level mine.

The unit of weight in all mining operations is the ton, which contains 2000 lbs. avoirdupois; in relation to this, the English ton of 2240 lbs. is spoken of as the "long ton."

According to the determinations by Mr. Franklin White, a block of 11.86 cubic feet of average Main Reef banket weighs one ton; so that in round figures 12 cubic feet may be considered as equal to one ton.

The assay values given throughout are in terms of fine gold per ton of 2000 lbs.; it is not usual to record the small amount of silver which occurs, except in assays of bullion.

The order of sequence employed in the latter chapters is as far as possible that actually followed in the exploitation of the ore and its delivery to the mill.

Four primary operations are taken to exist, viz.:—

Shaft-sinking.
Ore-developing.
Ore-stoping.
Ore-dressing.

The first three are performed underground and the latter on the surface. In the deep-level mines shaft-sinking is a much more important and distinct operation than it is in the outcrop mines, because, until the reef is reached, it proceeds alone and marks a definite stage in the history of the mine.

Around these primary operations are arranged those of hoisting, tramming, pumping, sampling, etc., the accounts of which are supplemented by descriptions of the various equipments.

Ventilation and a chapter on compressed air and machine drills are given at the end of the underground operations, and transport, including tramming, is placed between those operations and ore-dressing.

After the ore has been delivered to the mill, reduction is taken to commence. It includes all those operations, namely, milling with amalgamation, cyaniding, chlorination, slimes treatment, smelting, etc., during which gold is being extracted.

The operation of milling is often considered as coming within the range of ore-dressing, because the actual extraction of the gold does not commence till the ore has been reduced to a sufficiently fine state of division, but as some amalgamation takes place before the ore is delivered from the mortar-boxes of the mill, and as the two operations of milling and of amalgamation are economically inseparable, milling is considered as belonging to the processes of reduction.

CHAPTER II

GEOLOGY

BANKET is the name given by the Dutch to an auriferous conglomerate which, in its weathered appearance, resembles almond rock. It is known chiefly as occurring on the Witwatersrand Goldfields, although similar rock is found elsewhere in South Africa. This conglomerate occurs as beds intercalated in greater part through a quartzite formation, which is known as the Witwatersrand Beds. These beds, with others, appear to form two definite synclines, which may be called:

(1) The Witwatersrand Syncline.

(2) The Southern Heidelberg Syncline.

These synclines are best defined by the superficial extension and inclination of the horizon of the richer reefs, because these reefs have been so followed and opened up that the most complete information is available with regard to them. This horizon of the richer reefs includes those which form the Main Reef Series, and it lies always near the base of the Witwatersrand Beds, and often directly on the underlying quartzite-shale beds.

1. WITWATERSRAND SYNCLINE

Superficial Extension.—The extent of the horizon around this syncline is naturally divided into districts by breaks in the continuity of the outcrop, caused either by dislocation, as at the Witpoortje Break, or by concealment either under coal measures as at Springs, Vereeniging, and the south-western end of the long synclinal axis in the Orange River Colony,

or under the Dolomitic Formation between Randfontein and Klerksdorp.

These different districts are tabulated on page 10.

Central District: Roodepoort, Johannesburg, and Boksburg.—Starting from the Banket mine on the farm Roodepoort, about 13 miles east of Johannesburg, the Main Reef Series can be followed in a direction approximately east for a distance of about 27 miles to the property of the Blue Sky close to Boksburg. At that point the continuity on the surface is concealed by coal measures and obscured by igneous intrusions and overflows, but recent prospecting operations on the ground of the Boksburg Gold Mines and of the Apex Mines have shown the Main Reef Series to extend through these properties.

Van Ryn-Chimes District.—Three miles from the Blue Sky mine, in a direction E.N.E., the Main Reef Series has been undoubtedly picked up in the property of the Benoni Gold Mines, and from this point it can be followed, though somewhat broken in places, for about 6 miles in an E.N.E. direction on to the property of the Modderfontein Extension. Beyond this point it has been traced by bore-holes, and found to continue in much the same general course, though sinuous and broken in detail, for a farther distance of 4 miles across the northern half of the farm Klipfontein. Immediately beyond this point it has not been definitely traced, but there is every reason to suppose that it continues under the coal measures and surface deposits.

Nigel-Heidelberg District.—Fifteen miles from Klipfontein, in a direction S. by E., in the north-western corner of the farm Vlakfontein, the Nigel Reef, which belongs to the same horizon as the Main Reef Series, was struck in a bore-hole underneath a considerable depth of coal measures, from which it emerges in the property of the Marievale-Nigel. Here again there is every reason to suppose that it continues beyond this point under the coal measures in a

direction N.E. by N., more or less in line with its outcrop through the Nigel properties.

Granted, then, a continuation of the horizon from Klipfontein in the direction of strike E.N.E., and a continuation from Vlakfontein in the direction N.E. by N., these terminals should approach until, aided by the flattening of the dip which occurs in each case going eastwards, they should meet to form a continuous curve under the coal measures. The reef in the Vlakfontein bore-hole assures us of the extension of the Nigel Reef at least two miles on the other side of the Marievale-Nigel mine, whence it can be followed through the various Nigel properties, in a general south-west direction and for a distance of 10 miles, to a point on the farm Portje about 5 miles east of Heidelberg, where its outcrop becomes concealed under an igneous overflow.

Under this igneous overflow the horizon which is being followed would appear to cross the townlands of Heidelberg, and then to extend along the left bank of the Blesbok Spruit, till it reappears again on the farm Platkoppies, about 13 miles in a south-west direction from Heidelberg, where it has been opened up by the Platkop Estate Company, though it may be that the reef there opened up belongs to a lower horizon. Although hidden for the greatest part of this distance, the continuity of the horizon is proved by the continuous outcrop of the overlying banket beds, over which there is no igneous overflow.

Beyond Platkoppies the continuity is proved by the various outcrops of the overlying reefs; it follows a south-west direction on the left bank of the spruit, till at a point about 17 miles from Heidelberg the whole formation bends round with a south-easterly strike towards the Vaal River, where it becomes hidden under coal measures and igneous overflows, below which there is every reason to suppose that the horizon continues.

From the outcrop on Portje, before the reef becomes

concealed, to the point to which it can definitely be shown to extend, is a distance of 20 miles in a south-west direction.

Venterskroon District.—After concealment under the coal measures in the Orange River Colony the horizon again reappears at a point about 40 miles away in a westerly direction near the farm Wittekopfontein, close to the Vaal River, in the Venterskroon district.

Through this district there are evidences which make it likely that the horizon is coincident with the Acme, Red, and North Reefs. These reefs extend first in a general W.S.W. direction for about 16 miles, and then in a S.S.W. direction for about 12 miles, which brings them into the Orange River Colony near the town of Reitzburg.

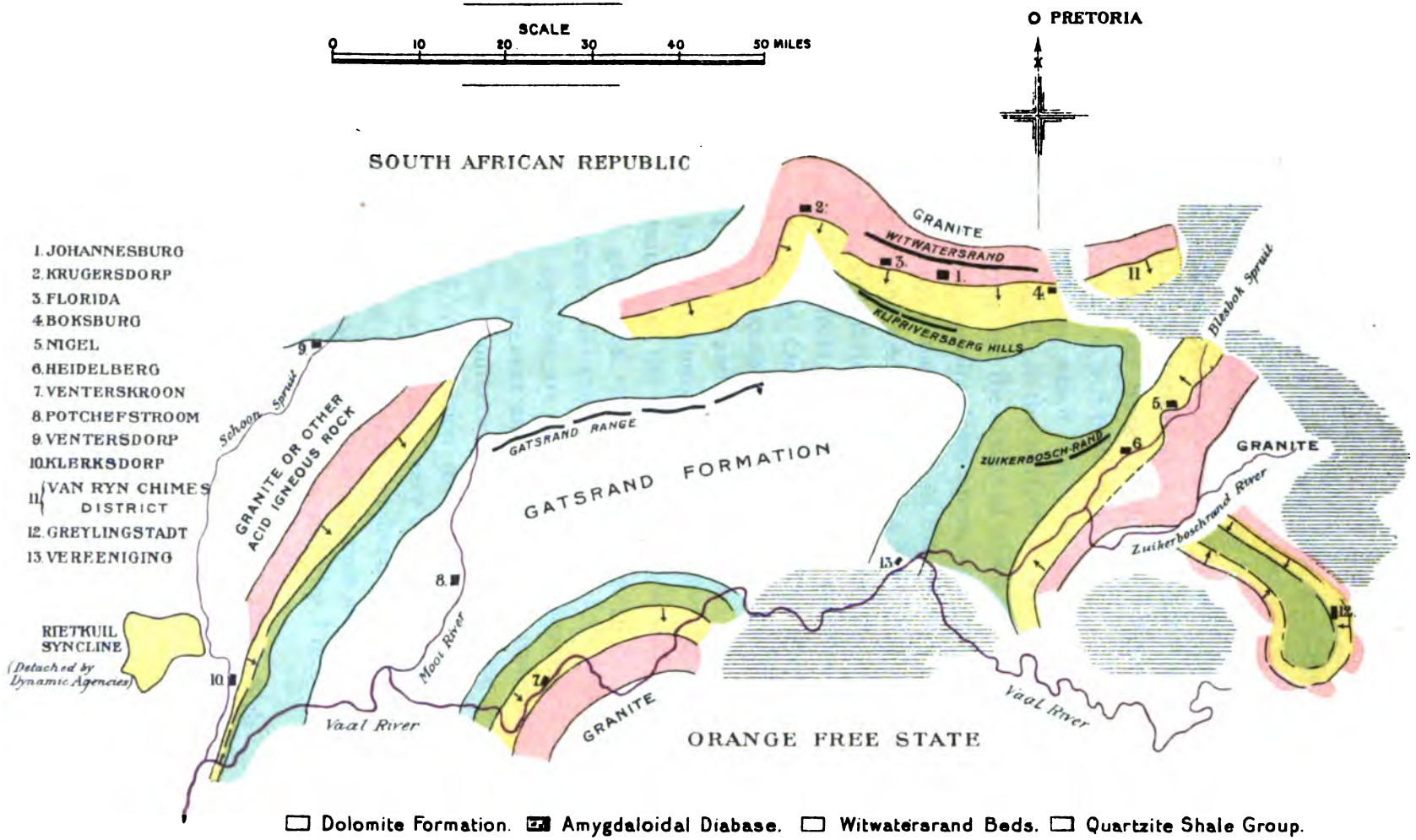
Orange River Colony.—For a short distance into the Orange River Colony the position of the horizon can be deduced and its continuity proved by reference to the outcrops of some of the overlying banket beds, after which it dips under the coal measures. The strike of these overlying beds suggests that the horizon does not extend far south of the Vaal River.

Klerksdorp District.—The horizon re-enters the Transvaal Colony again near the mouth of the Schoon Spruit, at which point it is about 34 miles away, in a W. by S. direction, from its passage into the Orange River Colony.

From here, through the Klerksdorp district to Buffelsdoorn, its position can only be approximately indicated, but the series of banket beds as a whole can be traced extending in a north-east direction. Between Buffelsdoorn and Randfontein the formation is greatly disturbed by igneous intrusions and concealed by later deposits, yet the general lie of the formation affords sufficient proof of the continuity of the horizon between these points.

Krugersdorp District.—From the southern boundary of the farm Randfontein, at a point about 8 miles S.S.W. from Krugersdorp, the Botha's Reef Series, which marks the horizon,

ROUGH MAP OF THE FORMATIONS WHICH EITHER CONFORM TO OR ARE FOUND IN, THE WITWATERSRAND AND SOUTHERN HEIDELBERG SYNCLINES.



To face Page 8

can be followed, practically in a continuous line to Grey's Mynpacht on Witpoortje, for a distance of 6½ miles in a N. by E. direction, and then for 15½ miles in an E. by S. direction.

Here the Botha's Reef Series is cut off by a fault called the "Witpoortje Break." Its termination on the other side of this fault is the Main Reef Series in the Banket mine, which is about 3 miles away in a S.S.W. direction.

Summary.—The horizon which has been followed has thus been proved to extend on the surface so as to form what is almost a closed traverse, the outline of which is roughly elliptical, the major axis extending from Springs to a point in the Orange River Colony, situated south of the centre of the line joining Klerksdorp and Venterskroon, in a N.E. by E. and S.W. by W. direction, and having a length of about 130 miles.

The greatest width is from Krugersdorp to a point under the coal measures in the Orange River Colony, situated roughly about 16 miles south of Vereeniging, a width of about 60 miles in a N.N.W. and S.S.E. direction.

These axes are therefore almost at right angles to each other.

The relative positions of the more important districts are shown in the rough geological map which faces page 8. In this map the distribution and extent of the various formations in the Witwatersrand Syncline are indicated with an approach to accuracy sufficient to prevent any wrong general conception being formed. The Southern Heidelberg Syncline is seen extending in a S.W. direction from Heidelberg; the various formations are there indicated very diagrammatically.

No detailed geological survey has yet been made of the country covered by this map; the latest attempts at mapping it out geologically have been made separately by Fred. H. Hatch and David Draper.

The horizon of the richer reefs is that outline from which the arrows, indicating the direction of dip, extend; it is 308 miles around, this aggregate distance being made up as follows:—

WITWATERSRAND SYNCLINE

Traverse of the Horizon of the Richer Reefs

District.	Position in District.	Direction of Strike.	Distance.	Angle of Dip.	Direction of Dip.	Continuity of Horizon.
			Miles.			
Central	Banket mine to Blue Sky mine	E.	27	50°	S.	Proved by outcrop
	Blue Sky mine to Benoni mine	E. by N.	3	...	...	Concealed by coal measures and broken by disturbance
Van Ryn-Chimes	Benoni to Modderfontein Extension	E.N.E.	6	52°	S.S.E.	Proved by outcrop
	Modderfontein Extension to Klipfontein	E.N.E.	4		S.S.E.	Proved by bore-holes
	Klipfontein to Vlakfontein	S.E.	15	...	...	Concealed by coal measures
Nigel-Heidelberg	Vlakfontein to Marievale-Nigel	S.W. by S.	2	26°	N.W. by W.	Proved by bore-hole
	Marievale-Nigel to Portje	S.W. by S.	10		N.W. by W.	Proved by outcrop
	Continuation down the Blesbok Spruit	S.W.	20	...	N.W.	Proved by outcrop of overlying beds
	From Heidelberg district to Venterskroon district	W.	40	...	...	Concealed under coal measures
Venterskroon	Wittekopfontein to Buffelskloop	W.S.W.	16	80°	S.S.E.	Proved by outcrop
	Buffelskloop to Kroomdraai at the Vaal River	S.S.W.	12		E.S.E.	Proved by outcrop
	Kroomdraai to Witkop in the Klerksdorp district	W. by S.	50	...	...	Concealed by coal measures; the distance given, 50 miles, is a rough estimate of its course under the coal measures
Klerksdorp	Witkop to the town of Klerksdorp	N.	8	40°	E.	Proved by outcrop
	Klerksdorp to Buffelsdoorn	N.E. by N.	14		S.E. by E.	Proved by outcrop
	Buffelsdoorn to Randfontein	N.E.	66	30°	S.E.	Half proved by disconnected outcrops of overlying reefs and half disturbed and concealed
Krugersdorp	Randfontein to Waterval near Krugersdorp	N. by E.	6½	65°	E. by S.	Proved by outcrop
	Waterval to Grey's Mynpacht	E. by S.	5½	33°	S. by W.	Proved by outcrop
	Grey's Mynpacht to Banket mine	S.S.W.	3	...	...	Broken by Witpoortje Break
			308			

The continuity is therefore proved by outcrops and borings for . .	164 miles
The continuity is concealed by more recent measures for . . .	123 ,,
The continuity may be considered to be interrupted by faults and dykes for	21 ,,
	308 miles

The large mileage of outcrop concealed by more recent measures is distributed over four principal areas, viz. at Springs, at Vereeniging, in the Orange River Colony, and from Buffelsdoorn to Randfontein. It has been stated that the evidences point to the continuity of the reefs under these measures; this is being continually confirmed in particular areas, and the length of mileage along which the continuity of the horizon is proved is being continually increased.

It should however be added that where the horizon of the reefs is concealed under more recent measures, *e.g.* at Vereeniging and in the Orange River Colony, it may possibly pass out of the syncline and extend beyond it.

Inclination.—At all points around the outline of this traverse, with the exception of that portion comprised in the Venterskroon district, the reefs at the horizon dip inwards towards the axes of the ellipse.

The different directions and amounts of dip in the different districts are shown in the statement on page 10; the directions are also indicated by arrows in the small map facing page 8.

Throughout the Central district the average dip at surface is about 50°, the range of inclination being from about 16°, which occurs at the Simmer and Jack, to an angle of nearly 90° in the Ferreira and Henry Nourse mines; the steepest dips are found in the neighbourhood of Johannesburg rather than at Roodepoort or Boksburg, as indicated by the angles of dip given with Fig. 29.

From this comparatively high average angle there is in

depth a pronounced flattening, which in the Central Rand is very marked. From the Langlaagte Estate on the west of Johannesburg to the Treasury on the east, the average dip on the surface is about 60°, and that in the deep-level shafts immediately to the south is about 31°, as seen from the following table :—

Langlaagte Deep	32°
Crown Deep	31°
Robinson Deep	36°
Village Main Reef	30°
Wolhuter Deep Shaft	25°
Nourse Deep	33°
Jumpers Deep	28½°
Average	31°

This flattening is well illustrated in the shafts of many of the outcrop mines. The following statement of average dips was made from the cross-sections through the Ferreira shafts :—

DIPS IN THE FERREIRA MINE

Levels.	Dip.
Surface to 120 ft. level	87°
120 ft. level to 220 ft. level	84°
220 „ 320 „	72°
320 „ 420 „	68°
420 „ 520 „	55°
520 „ 620 „	49°
620 „ 720 „	38°
720 „ 820 „	36°
820 „ 920 „	31½°
920 „ 1120 „	34°

In addition to this flattening in depth along the plane of the horizon, there is also in most cases a noticeable flattening along the surface from the outline of the horizon, towards the axes. Thus, to the south of Johannesburg the dip of the outcrop of the Main Reef Series at the Ferreira Mine is 87°, whereas that of the Bird Reef Series is about 34°, between which two dips it is not suggested that any unconformity

exists. Farther to the south the Kimberley and Elsburg Series outcrop at their respective distances with a dip of just over 30°, and still farther south, about 7½ miles from Johannesburg, the Black Creep outcrops with a dip of about 18°, though between this reef and the older banket series there is undoubted unconformity.

Though this flattening from the outcrop is general, there are one or two exceptions. The reef in the Vogelstruis Estate gets steeper below, the increase in dip being greatest in the West Incline shaft, where, at an incline depth of 580 feet from the surface, the South Reef dips about 58°, whereas at surface the dip is about 45°; this increase in dip is confirmed by the inclination of the strata in the shafts of the Vogelstruis Deep.

In the central portion of the Simmer and Jack a pronounced steepening of the reefs sets in after a short flattening from the surface; this is seen from the following statement of dips taken from No. 6 winze in that mine:—

3rd level	to	4th level	. . .	10° 40′
4th ,,		5th ,,	. . .	7° 30′
5th ,,		6th ,,	. . .	5° 20′
6th ,,		7th ,,	. . .	0° 30′
7th ,,		8th ,,	. . .	18° 50′
8th ,,		9th ,,	. . .	26° 10′
9th ,,		10th ,,	. . .	26° 25′
10th ,,		12th ,,	. . .	37° 00′
14th ,,		16th ,,	. . .	29° 35′

It would appear that this secondary steepening has an extended occurrence along the Witwatersrand, because it is now found that in many cases the reefs have flattened from the outcrop down to an angle, which is often less than that at which they have been found to be lying in the shafts of the deep-level properties immediately to the south. This important fact is illustrated in the following table:—

TABLE OF DIPS SHOWING THE GENERAL FLATTENING OF THE REEFS FROM THE OUTCROP AND THE SECONDARY STEEPENING IN GREATER DEPTH

Outcrop Company.	Dip at Surface.	Dip to which the Reefs have flattened.	Level of Low Dip.	Deep Level Company.	Distances of Shafts from Outcrop.	Dip.	Position at which Dip was taken.
					Feet.		
Vogelstruis Estate .	45°			Vogelstruis Deep, west shaft	2190	50°	Strata in shaft
Crown Reef . .	51°	30°	7th to 8th level	Crown Deep, 1 and 2 shafts	1550 and 2370	31°	1st to 3rd level
Robinson, East Incline	65°	32°	9th to 11th ,,	Robinson Deep, No. 2 shaft	3200	33°	Reef in shaft
Ferreira . . .	87°	31½°	8th to 9th ,,	,, ,, No. 1 ,,	4060	38°	,, ,,
Jumpers East Shaft .	60°	23°	5th ,,	Jumpers Deep, No. 1 shaft .	1630	28½°	,, ,,
Geldenhuis Estate Main Incline . . .	30°	0°	9th ,,	Geldenhuis Deep, No. 2 shaft	2070	22°	,, ,,
Simmer and Jack—				Simmer and Jack—			
No. 3 winze . .	4½°	0°	3rd ,,	No. 2 deep level shaft .	2410	40°	,, ,,
No. 8 ,, . .	18°	5°	3rd to 4th ,,	No. 1 ,, ,, .	1950	46°	,, ,,
East Incline . .	27°			No. 3 ,, ,, .	1060	42°	,, ,,

Further, the No. 2 Incline shaft of the Langlaagte Deep, which was started down on the reef at an angle of 31°, was, after the first 200 feet, changed to 34° to follow the steepening of the reef.

A good instance of secondary increase of dip in depth is shown in the East Incline shaft of the Geldenhuis Deep, where the following dips were recorded :—

From 2nd level to 3rd level . . .	23° 25′
„ 3rd „ 4th „ . . .	24° 15′
„ 4th „ 5th „ . . .	26° 50′
„ 5th „ 6th „ . . .	28° 45′

The trend of this secondary steepening can be followed from the Langlaagte Deep to run almost parallel with the formation and to cross the outcrop at the Simmer and Jack.

From the following table there would appear to be another similar but deeper occurrence between the Simmer East and the May Consolidated :—

Company.	Dip.	Horizontal Distance from Outcrop.
May Consolidated .	At outcrop, 72° .	Nil.
„ „ .	At bottom, 40° .	500 feet.
Glen Deep No. 1 shaft	. . 30° .	1820 „
Simmer East Lohse shaft	. . 40°[1] .	5080 „

Secondary steepening, as met with up to the present, has the nature of gentle undulation; as it is accompanied by a compensating flattening on either side, it will not prevent the reefs from becoming horizontal in depth. This, however, does not imply any extended horizontal area, because it is quite possible that the reefs may start at once to rise again in the opposite direction.

In the Van Ryn and Chimes district the average dip on surface is about 52° in a S.S.E. direction, the range of inclination being from 25° in the Chimes West to 90° in the Chimes.

[1] As determined from drill core.

In this district there is also undoubted flattening from the surface, as is seen from the following dips taken in the Van Ryn West :—

No. 1 Mine: Main Reef

Surface to 1st level	66°
1st level to 2nd level . . .	48°

No. 2 Mine: Main Reef

Surface to 1st level	74°
1st level to 2nd level . . .	68°

No. 4 Mine: Chimes Reef

Surface to 1st level	63°
1st level to 2nd level . . .	50°
2nd „ 3rd „ . . .	45°

In the Nigel district the average dip of the reefs at surface is 26° in an average direction N.W. by W. In the Nigel Deep "D" shaft the dip is 8°, and in "C" shaft 10°, whereas immediately to the outcrop of these two shafts it is about 15°. In the Venterskroon district the reefs dip about 70° in a south-east direction, away from the axes of the closed traverse. It seems likely, in view of the proximity of the granite, that these beds are overthrown and that their true dip should be to the north.

In the Klerksdorp district the average dip is 40° in a direction south of east, and in the Krugersdorp district the reefs at the horizon which is being followed, dip at an average angle of 36° towards the axes. In these last two districts the flattening from the outcrop, whilst not pronounced, is nevertheless often to be observed.

From the extension of the horizon of the richer reefs on the surface to form what is almost a closed traverse, and from its extension in depth as a flattening curve, it is seen that, in relation to a main axis, it takes a form of a syncline, and just as, in completing the circuit, its extent on the surface can be followed continuously from one strike, through all the inter-

mediate directions, to another quite opposite, so there would appear to be little doubt that its extent in depth is continuous from a dip to the south on the north side, to a dip to the north on the south side.

2. The Southern Heidelberg Syncline

In this district the same horizon—that of the richer reefs occurring near the base of the Witwatersrand Beds—may in the same way be followed. From the farm Rietfontein, about 13 miles S.E. from Heidelberg, it can be traced by disconnected outcrops to extend to Doornhoek, a distance of 15 miles in a direction S.E. by E. From this point to the Heidelberg-Roodepoort mine, a distance of 15 miles in a S.S.E. direction, the continuity is broken. At this mine the outcrop strikes about S. by E., but does not extend far.

At a point 10 miles from this mine in a S.S.W. direction the horizon is again picked up on the farm Hex River. From this point it strikes east, and then passes through the farms Barnards Kop, Hartebeestefontein, Roodeval and Rietvlei in a N.E. and then in a N. by E. direction. This outcrop, though broken and disconnected, proves the continuity for 15 miles.

Between Rietvlei and Rietfontein, which lies about 7 miles west of Greylingstadt, there appears to be a break. From this latter farm to Malans Kraal, a distance of 15 miles, the outcrop of banket reefs is continuous, extending in a general direction W.N.W.

At this point the series abruptly ends, probably against a fault, which striking N.E. by E. would extend to the outcrop on the farm Rietfontein, the starting-point used in tracing this syncline. It is along this fault that the Southern Heidelberg Syncline appears to be severed, and separated by a downthrow, from the Witwatersrand Syncline.

The longer axis of the former syncline extends for about 24 miles in a direction S.E. by E.; those outcrops which are on the

north-east side have an average dip of about 47° towards the longer axis, and those on the south-west side dip about 42° towards the same axis. There would thus appear to be no doubt that the banket beds are arranged around this axis as a synclinal fold, so that in depth they will be continuous, and having a flattening dip, they will at some point be found to be horizontal. A section across this syncline is shown in Fig. 12.

At about the centre of the long axis this syncline has its least width of about 6 miles, viz. from the farm Wilgepoort on the south-west side to Rietbult on the north-east; on either side of this line the width increases, till, near each end, the syncline is about 10 miles across. At the north-western end the horizon is not continuous because of the fault mentioned above, and at the other end, disconnected outcrops mark the line, along which there was once continuity, but which has subsequently been shattered by igneous intrusions.

Geological Structure

The following formations, placed in ascending order of sequence, either conform to the synclines, which have been demonstrated to exist, or are found in them :—

Granite.
Quartzite-Shale Group.
Witwatersrand Beds.
Amygdaloidal Diabase.
Black Reef Formation.
Dolomite Formation.
Gatsrand Formation.
Coal Measures.

The Granite.—This term embraces the whole occurrence of allied igneous rocks which are found as an envelope to the formations that are situated within the synclines.

It occurs most prominently to the north of Johannesburg, where its outcrop is about 17 miles across. This occurrence

extends, east and west, parallel with the northern side of the syncline, for considerable distances, as shown in the map facing page 8.

There is another strong outcrop centred at Vredefort in the Orange River Colony; it is this occurrence which has caused the excessive tilting in the Venterskroon district. Though at this place its extension is more like an intrusive boss, it appears in part forming a ridge running parallel to the bedded formation.

At a point about 13 miles due east of Heidelberg, and again between Randfontein and Buffelsdoorn, there are other outcrops arranged around the periphery of the syncline.

At a point due west of Klerksdorp there is an outcrop of an acid igneous rock which probably belongs to the same formation.

Although the granite cannot actually be traced right round the syncline, there is no doubt that it forms a complete envelope for the formations under consideration. In many places, on account of the amount of disintegration it has suffered, it is now hidden under more recent measures, whilst the quartzites of the overlying formations, having resisted denudation, have not been covered up.

Along the south-western extension of the Heidelberg Syncline the granite was recently struck underneath extensive later deposits, in a bore-hole put down by the South Rand Gold Corporation. All the occurrences of granite which have been mentioned extend in conformity with the syncline. At a point about 15 miles south-west from Randfontein there would appear to be a departure from this regularity, for whilst the main mass of granite is in its proper position, an arm, extending from it, strikes across the bedded formation inside the syncline.

Whilst granite is used as an inclusive term for the basement rock, there are varieties in the mass which may more definitely be described as granulite, felsite, syenite, etc., and at its contact with the sedimentary rocks gneiss and schists

have resulted from its deformation. This is especially the case along the occurrence north of Johannesburg, the gneiss and schists at that place being over a mile across.

Quartzite-Shale Group.—This group of hard quartzites and ferruginous shales occurs lying on the granite.

The order of succession of the component beds which is maintained at Johannesburg and Venterskroon is as follows: there is first a bed of quartzite resting immediately on the schists; then a series of alternate bands of highly ferruginous shales and hard quartzites, in which the shales predominate; and then a smaller thickness of hard quartzites with some small bands of shales and sandstones.

North of Johannesburg the first quartzite bed is 5000 feet wide on the surface, and it dips to the south at angles varying from 20° to 50°; the next series of quartzites and shales is about 9600 feet across, the overlying 1000 feet being that bed of highly ferruginous shales so well known as the Hospital Hill Shales, which occupies the highest point of the Witwatersrand range of hills. The top bed of quartzites dips to the south at an angle of from 40° to 50°, and is about 1600 feet across. The total horizontal thickness of this group north of Johannesburg is thus about 16,200 feet.

Though this sequence of quartzites and shales is generally maintained, the thickness of the beds and the dip may both vary considerably within short distances.

At Venterskroon the whole series, which dips 70° or 80° to the south, has a horizontal thickness of about 16,000 feet. At this place, interstratified with the shales, there are considerable thicknesses of diabase; and at the base of the series, lying on the granite, there is a peculiar igneous rock, described as a much-altered trap rock.

This Quartzite-Shale group outcrops all around the Witwatersrand and Heidelberg Synclines. It is well exposed on the north flank of the granite boss, east of Heidelberg, where it forms high ridges, extending in a south-west direction

parallel with the Blesbok Spruit till they pass under the coal measures near the Suikerbosch Rand River.

The greatest extent of this group is north of Krugersdorp, where, from its position on the gneiss and schist, it extends south for about 30,000 feet, to the base of the Witwatersrand Beds, quartzites and shales alternating across this distance. Auriferous banket occurs in this series, generally along those contacts of quartzites and shales which have quartzite for the hanging wall and shale for the foot wall.

These banket beds are rarely very thick, but they are very persistent, being traceable over great lengths; though they are generally found at these contacts they sometimes leave them and are found, in limited extent, within the quartzite itself.

In the lowest bed of quartzite there is banket; on the Houghton Estate, a property lying north-west from Johannesburg, there is a narrow conglomerate band quite close up against the schists and gneiss.

The Du Preez Reef Series, to which belong the banket beds that are being worked on the farm Rietfontein, north of the Witwatersrand mine, about 9 miles E.N.E. from Johannesburg, occurs in this lower bed of quartzite, though, here too, the most northerly banket bed, having a quartzite hanging wall, rests on shale.

In the Heidelberg district a series of these contact reefs occurs, of which the Molyneux Reef is the overlying one. This occurrence of banket can be followed from the farm Holgatfontein, about 4 miles east of Nigel, parallel with the Blesbok Spruit, to the farm Elandsfontein, about 10 miles in a S.S.W. direction from Heidelberg. In this Quartzite-Shale group at Venterskroon also, small layers of conglomerate are found, and the quartzite itself appears in places as a very coarse grit.

From Klerksdorp to Randfontein there are several occurrences of banket beds in this group, viz. on the farm Witrandgesfontein and on the farm Modderfontein; at this latter

place the quartzites and shales, with some conglomerates, are found resting in order upon the granite. Nearer Randfontein also, on the farms Doornfontein and Witfontein, there are banket beds in the quartzites of this group. North of Krugersdorp one of the banket beds found in this series is the Government Reef, and still farther to the north there is another, the Alexandra Estate Reef. East of Krugersdorp, in a section across the formation at Florida, several banket beds occur, which can be recognised as being very similar to the Du Preez Series.

These occurrences demonstrate that auriferous banket beds occur in the Quartzite-Shale group.

The extent of this group between the overlying banket formation and the underlying schists varies considerably. North of Johannesburg, with two main beds of shale and a dip of 45°, it is, as previously stated, some 16,200 feet across on the surface; north of Krugersdorp, with five main beds of shale, more or less, and a dip of 25°, it is 30,000 feet across; whilst farther to the south-west, towards Klerksdorp, with five main beds of shale, more or less, and a dip of 30°, it is about 35,000 feet across on the surface. In the Heidelberg district, as mentioned above, there are at least three beds of shale dipping about 20°; and against the Vredefort granite, in the Venterskroon district, this group, dipping 70° to 80°, is about 16,000 feet across on the surface.

The shale in these last two districts is often a very hard rock, which might be either a shale very much indurated and altered, or a trap rock.

The outcrops of the quartzites of this group are often irregular, contorted and disturbed, and the structure of both quartzite and shale shows evidence of considerable deformation. The banket has often a burnt appearance, quite distinct from the clear and vitreous look of the overlying banket of the Witwatersrand Beds, and the pebbles are generally small and often crushed and elongated.

On the farm Rietbult, situated on the north-east side of the Heidelberg Syncline, an auriferous banket bed occurs in this Quartzite-Shale group, as shown in the section, Fig. 12.

The Witwatersrand Beds.—These are quartzites with beds of conglomerates, sandstones and shales, which extend from the Quartzite-Shale group to the Amygdaloidal Diabase. This series lies inside the Witwatersrand and Southern Heidelberg Synclines, and conforms to their outlines.

Its stratigraphical relation to the underlying beds is not quite clear. At Johannesburg its base is dipping at a higher angle than the Quartzite-Shale group, the average dip from the Crown Reef to the City and Suburban being 65°, whereas the underlying beds dip at about 45°; between the two formations there is, however, evidence of faulting.

At Rietfontein the Du Preez Reef of the Quartzite-Shale group dips 55° at the surface, and the average dip of the North and South Reefs of the Main Reef Series on the Witwatersrand property is 54°, so here the dips are practically conformable.

In the Nigel district, from the granite, across the formation, the dip gradually decreases, so that the Quartzite-Shale group, being nearer the granite, dips more steeply than the overlying banket formation.

In the Venterskroon district the two formations are apparently conformable; they both dip out of the syncline and, by doing so, form an exception to the general rule.

From Klerksdorp to Randfontein, and at Krugersdorp, these beds are apparently conformable; at Florida the quartzites and shales dip about 80°; going south across the formation at this point, this high angle gradually decreases till at the Main Reef on the Bantjes the dip is 35°.

Generally, then, it would appear that these overlying beds are conformable to the Quartzite-Shale group, and in the cases where unconformity apparently exists, the relation between the two formations is probably a faulted one.

The horizontal thickness of the outcrop of these beds across the formation varies considerably; south of Johannesburg this thickness is about 26,400 feet, which, without reverse faulting and at a dip of 30°, is equal to a true thickness of over 15,100 feet; in the Nigel district the horizontal thickness is about 19,800 feet, and the true thickness about 8700 feet, the average dip being 26°; at Venterskroon the horizontal thickness is about 12,000 feet, and the true thickness 11,800 feet, the dip being 80°; from Klerksdorp to Randfontein the formation is much disturbed; at Krugersdorp the horizontal thickness is about 9000 feet, and the true thickness about 3800 feet, the dip being 25°; at Rietbult, on the Heidelberg Syncline, the horizontal thickness is about 2500 feet, and the dip being 60°, the true thickness is about 4300 feet; at the Heidelberg-Roodepoort mine the horizontal thickness is 3000 feet, and, the dip being 33°, the true thickness is about 1620 feet.

The great diversity in these figures makes it likely that the outcrop of this formation affords no real guide to its true extent in different parts.

There are reverse faults along the Witwatersrand which, in upthrowing the reef on the south side, sometimes amounting to duplication on surface, increase the outcrop of the whole series. There are also evidences which show that the amygdaloidal diabase does not mark the upper limit of this formation, but that, as near the Heidelberg-Roodepoort mine, it encroaches on it, so as to decrease the width of its outcrop.

It would, however, be incorrect to suppose that by the application of these corrections, all the thicknesses at different points could be brought to correspond to within a few feet of one another, for reference to the distances apart of the various banket beds shows that in undisturbed sections the distances between the same reefs vary.

It may be also that in places where the formation appears unduly narrow, a downthrow on the dip side may

have occurred which, having the opposite effect to an upthrow, would decrease the width of the outcrop.

In this formation the following banket series occur :—

1. The Main Reef Series.
2. The Livingstone Reef Series.
3. The Bird Reef Series.
4. The Kimberley Series.
5. The Elsburg Series.

A more detailed description of the above series is given in the following chapter.

The quartzites are usually compact, hard, and, owing to the presence of some finely disseminated unoxidised pyrites and dark silicates, of a dark colour with a greenish tinge, known locally as "blue." In places they are gritty, passing sometimes imperceptibly into fine banket, and in other places they are very micaceous, this being especially noticeable near the Bird Reef. In this large extent of quartzites there is usually very little interstratified shale; closely underlying the Kimberley Series there is generally, however, a characteristic occurrence of this rock which is often calcareous. Along the Central Rand this bed is not large; it increases going eastwards round the syncline, and in the Nigel district and at places around the Heidelberg Syncline it is often 400 feet thick.

Amygdaloidal Diabase.—Overlying the Witwatersrand Bed there is a large occurrence of igneous rock, which, running parallel with the formation, forms to the south of Johannesburg the range of hills known as the Klipriversberg Range.

Across the formation south of Johannesburg, this rock is some 19,000 feet thick, measured on the surface; it would there appear to lie conformably with the formation, though the nature of the junction is uncertain.

In the Nigel district all the various reef series, as tabulated above, are found outcropping, so that in this

district this diabase occupies a position similar to that south of Johannesburg. On the farm Spaarwater, about five miles north of Heidelberg, the thickness of the diabase would appear to be considerably less than it is south of Johannesburg, as it cannot be more than about 5000 feet across; going, however, in a south-west direction, it increases considerably in superficial extent, and occupies the northern slope of the Suikerboschrand range of hills; it does not, however, continue in this direction long, as it is not met with along the line of section from Johannesburg to Vereeniging except at the Klipriversberg.

In the Venterskroon district this diabase also occurs; the reefs, which outcrop here, have been correlated with the Main, Bird, and Kimberley Series, in which case the diabase must cover the outcrop of the Elsburg Series. In a section N.E. and S.W. through the farm Leewfontein, about 5 miles north-east from Venterskroon (see p. 56), this rock is shown to have a horizontal thickness across the formation of 18,600 feet. Around Klerksdorp, sheets of this diabase cover large areas of the banket formation unconformably; the upper limit to its extent is regularly contoured by the outcrop of the Black Reef, but the lower limit is irregular, and it would appear to interfere with, and encroach upon, the Witwatersrand Beds below it. In this district it first appears in position under the Black Reef on the farm Witkop, due south from Klerksdorp, just near the Vaal River, and after ramifying amongst the older conglomerate beds at Klerksdorp, it continues more regularly in an E.N.E. direction as a bed underneath the Black Reef, with an average horizontal thickness across the formation of about 6000 feet. On the farm Wildbeestlaagte, about 30 miles from Klerksdorp, it thins out, and around the Witwatersrand Syncline it does not occur again till the farms Rietvlei and Luipaardsvlei, which adjoin Krugersdorp, are reached; on the latter there is a small outcrop of it, which occurs a short distance south of that

series of conglomerate beds which includes, as its lowest layer, the Battery Reef of the West Rand. There is sufficient evidence to correlate this reef series with the Kimberley Reef of the Central Rand, in which case the Elsburg Reef lies concealed by the diabase.

Farther east, round the syncline, the diabase outcrops again near the north-eastern corner of the farm Doornkop, about 10 miles due south from Krugersdorp, from which point it opens out like a wedge to form the Klipriversberg Range.

In the Heidelberg Syncline this diabase occupies, on the surface, the whole area within the outcrop of the banket beds, with the exception of some small patches occupied by coal measures. Its superficial width in places must be fully 30,000 feet, and it has an average width of at least 24,000 feet.

In the south-eastern corner of the farm Rietbult the various banket beds which outcrop have been correlated with the Main Reef, Bird Reef, and Kimberley Series, and the diabase covers the position which should be occupied by the Elsburg Series; farther to the south-east on the same farm, the whole series of banket reefs is hidden under diabase.

Across the formation at the Heidelberg-Roodepoort mine the diabase occurs about 1500 feet, more or less, to the dip of the reef outcrop; this reef has characteristics of occurrence and formation which render its correlation with the Main Reef Series more than highly probable; in the short distance from the reef outcrop to the occurrence of the diabase, it is not conceivable that the Elsburg Series has space to occur, so that it must be looked upon as being covered up by the diabase.

In the northern corner of the farm Dassport, about 7 miles south from Rietbult, on the other side of the syncline, a banket reef occurs high up in the hills, right in the middle

of the diabase, striking and dipping conformably with other underlying banket beds, which outcrop a short distance to the south; in this case the diabase between the two must be lying on the upturned edges of the banket series. These statements would show that the diabase cannot be taken as marking the upper limit of the older conglomerate series, but rather that it overflowed its denuded outcrop. This amygdaloidal diabase has been described by Professor Judd as "a scoriaceous amygdaloid, probably basic," and by Hatch and Chalmers as "a hard fine-grained greenstone or melaphyre." Molengraff says: "The deposition of the Witwatersrand beds was followed by an overflow of basic eruptive rocks, diabase, melaphyr and amygdaloid, as well as diabase and porphyries, which formed a covering of considerable extent and thickness."

Wilson Moore says in his paper, "Some South African Rocks, chiefly Igneous," read before the South African Association of Engineers and Architects, 31st October 1894: "In my own examination, the point which strikes me most is the apparent infiltration of the quartz into the amygdules accompanied by titanite, which does not appear to have been present in the original mass, evidently a porphyry of plagioclase. The peculiar aggregation of these crystals (of titanite) in regular form, corresponding to the outline of the amygdules they fill, looks as if there has been some rotary movement in the material filling them while cooling, and I think that this is in part borne out by the position of the altered hornblende crystals in another."

Denny, in his book *The Klerksdorp Goldfields*, p. 20, says: "In its contact with the reef just mentioned (Black Reef) the diabase presents many phases of alternative action. In places it is laminated, greenish in colour, and quite steatitic; at other points massive and aphanitic; in a third, as at the deeper workings on the Eleazer farm, typically brecciated; and lastly, especially in the region of the East-

leigh mines, amygdaloidal to a high degree. . . . The amygdules consist principally of spherules of carbonate of lime."

A bore-hole put down by the Orion Company on their property was still in the amygdaloid at a depth of 2300 feet, after having encountered this rock at a shallow depth from surface.

The Black Reef Formation.—This formation overlies the diabase; briefly described, it consists of an underlying reef of the banket type, from a few inches to 14 feet in thickness, separated from the diabase by a few inches of dark ferruginous clay, and having immediately above it a series of alternating quartzites and shales, from 30 to 75 feet thick. Overlying these again there is a bed of quartzite varying from 10 to 100 feet thick; this constitutes the entire formation, which in turn is overlain by dolomite. From its outcrop near the Orion mine, where it dips about 20° to the south, it has been followed in an easterly direction for about 11 miles, and then in a general S.S.E. direction for about 8 miles to the farm Spaarwater, situated about 5 miles north of Heidelberg, where its outcrop is only about 5 miles to the dip of the Nigel Reef; from this point it continues, in irregular course, along the north-eastern slope of the Suikerbosch range of hills, down to a point east of Vereeniging, where it passes under the coal measures. Westward of this point its outcrop is not marked till the Venterskroon district is reached.

In the line of section through Johannesburg to Vereeniging, the only outcrop of the Black Reef occurring, is that just south of the Klipriversberg Range. In a section across the formation at Venterskroon, the Black Reef formation is found situated about 30,000 feet to the dip of the horizon of the richer reefs.

At Klerksdorp this formation has received its greatest development, and there the reef is called the Boschrand

Reef. Its first occurrence is on the farm Witkop, just north of the Vaal River and due south of Klerksdorp; from this place its outcrop can be traced for about 38 miles to the north boundary of the farm Welgegund, first in an E.N.E. and then in a N.E. direction. Along this length the rocks of this series have a dip of about 10° in a direction E. by S., and apparently they are unconformably overlain by the limestone formation, which lies horizontally, and underlain by the amygdaloidal diabase.

From this point, a broken outcrop can be followed for a distance of about 20 miles in a general direction E.N.E. on to the farm Varkenskraal.

Along the greater part of this distance the Black Reef rests immediately on the Witwatersrand Beds, and in places the unconformity between the two is well expressed, the Black Reef formation dipping about 10° and the underlying formation 30°. At this point, on the farm Varkenskraal, the Black Reef formation crosses right over on to the granite, the strike of its outcrop changing to a S.W. by W. and the dip to a N.W. by N. direction; this course is maintained for about 5 miles, beyond which, the northern outcrop, diverging from the southern, strikes in a more westerly direction and can be traced to Ventersdorp, thus arriving outside of the Witwatersrand Syncline.

Continuing east from Varkenskraal around the syncline, the formation outcrops again on the farm Rooipan, about 5 miles farther on, where again there are diverging north and south lines of outcrop, of which the north may be traced beyond the farm Kromdraai, situated 8 miles north of Krugersdorp, and the south, for a length of about 10 miles, to the western boundary of Wildefontein, where it would appear to be faulted by a granite dyke.

East of this dyke the Black Reef formation reappears again on the western boundary of the farm Blaawbank, whence it can be traced on to the farms Middlevlei and Lui-

paardsvlei, situated about 15 miles in a southerly direction from Krugersdorp; beyond this there are several outcrops in a disturbed country where the beds occur dipping and striking in all directions; along this length, from the farm Blaawbank, the Black Reef rests in greater part directly on the Witwatersrand Beds; at Doornkop, about 12 miles south of Krugersdorp, it appears again in a settled position, from which it can be traced continuously back to the Orion mine.

The Black Reef formation inside the Witwatersrand Syncline is thus connected with that occurring outside, by a neck extending over the encircling granite on the farms Varkenskraal and Rooipan, which are situated about one-third of the way from Krugersdorp to Klerksdorp.

Inside of the syncline the outcrop of the Black Reef formation roughly follows that of the older beds, though in places it diverges considerably from it.

Near Heidelberg the Black Reef is about 5 miles to the dip of the Nigel Reef; going S.W. from here the two reefs diverge, so that across the formation at Platkoppies, 15 miles away, the distance between them is about 16 miles. South of Johannesburg this distance is about 7½ miles, whilst at some point on the west side of the syncline the Black Reef must lie directly on the horizon of the Main Reef.

A section, north and south from the Orion mine to the southern outcrop of the Black Reef, made from bore-holes, shows that along this line the Black Reef at the deepest is just over 800 feet below the surface; assuming such depth to be at the centre of this section, the average dip to the outcrops on either side would be about 2°.

In the lower levels of the Orion mine the dip has flattened very considerably, and bore-holes placed immediately south of this mine struck the reef at depths which make the average dip to the outcrop to be from 3° to 6°.

In the Heidelberg Syncline there is no occurrence of Black Reef, it having presumably all been denuded away.

The alternating character of the quartzites and shales lying above the Black Reef is shown by the following section obtained in a bore-hole on Luipaardsvlei at a point about 15 miles in a southerly direction from Krugersdorp.

Surface soil and quartzite	190	feet.
Shale	42½	"
Quartzite	6	"
Shale	4	"
Quartzite	1	"
Shale	5	"
Quartzite	1½	"
Shale	1½	"
Quartzite	¼	"
Shale	2	"
Quartzite	2	"
Shale	4	"
Quartzite	2	"
Black Reef	...	
Shale	2	"

The Dolomite Formation.—This formation consists of dolomite with some bands of hornstone or chert in the lower portion, and of hornstone or chert with some dolomite in the upper portion; with the dolomite there are occasionally limited occurrences of earthy manganiferous and other metallic ores. Both the dolomite and the hornstone are of a dark blue or black colour.

Inside of the Witwatersrand Syncline the lower limit of this formation follows the outline of the Black Reef, and this relation holds good outside also, wherever that reef can be recognised. Generally this formation would appear to lie conformably with the Black Reef, so that apparently a comparatively very little interval of time would have elapsed between the deposition of the two formations.

The dolomite, like the Black Reef, generally lies at a low angle of dip; it is rarely steeply inclined, and where this happens to be the case, it is due to local faulting or intrusion of igneous rock. South of Johannesburg this formation has a

width on the surface of about 11,450 feet; in the same line of section continued beyond the Gatsrand quartzites to Vereeniging, the width exposed on the other side of the syncline is about 55,450 feet, and the dolomite still continues under the coal measures. This great apparent thickness of the dolomite is due to the obliquity of the line of section with reference to the course of the formation in that part of the syncline.

In the Venterskroon district the width on the surface is about 5600 feet. On the north-west confine of the syncline, where the Black Reef formation bursts out over the anticlinal axis of the granite, the superficial extent of the dolomite is of course much greater.

The Gatsrand Formation.—This consists of a series of quartzites and shales which occupy the axis of the Witwatersrand Syncline, forming the Gatsrand range of hills, which runs from a point about 12 miles south of Johannesburg, almost uninterruptedly, to near Frederickstadt, a station on the Klerksdorp line, a distance of about 55 miles in a W. by S. direction.

In a section from Johannesburg to Vereeniging this formation has a surface width of about 76,750 feet; for about four-fifths of this distance, on the northern side, the dip is 20° to the south, and for the remaining portion it is about the same angle to the north. This series forms the underlying beds between Klerksdorp and Venterskroon; but towards the other end of the longer axis of the syncline it does not extend far beyond the line joining Johannesburg to Vereeniging.

The quartzites are hard and dense; south of Johannesburg the shales occupy the greater portion of this series, the whole of which has been greatly disturbed. Overlying the dolomite, and probably coming between it and the Gatsrand quartzite, there are one or two occurrences of a conglomerate, which includes large pieces of diabase, dolomite, quartzite, etc., and is quite different from any of the Witwatersrand bankets, being probably much younger.

The nature of the stratigraphical relation of this series to the underlying dolomite is uncertain; generally it would appear to be conformable, though instances of unconformity occur. It dips at much the same angle as the dolomite. The whole formation bears evidences of considerable alteration, and it is quite likely that some of the shales are weathered outcrops of igneous intrusions.

Coal Measures.—This formation, occurring principally outside of the syncline, is found at places overlapping it; it covers up the eastern and western extremities of the longer axis of the Witwatersrand Syncline, at Springs and near Rietzburg in the Orange River Colony respectively, as well as the southern end of the shorter axis at Vereeniging; it also occurs in the south-eastern end of the Heidelberg Syncline. At all of these places coal beds have been worked.

This formation lies horizontally on the upturned edges of the lower beds.

Summary

The following tabulated statements form a summary of the superficial extent of the various beds:—

Witwatersrand Syncline

The Quartzite-Shale Group

	Width of Outcrop. Feet.	Average Dip. Degrees.	Calculated Thickness. Feet.
At Johannesburg . .	19,200	45	13,440
„ Nigel . . .	36,000	35	20,300
„ Venterskroon . .	16,430	80	16,000
„ Krugersdorp . .	30,000	30	15,000
„ Klerksdorp . .	disturbed		

The Witwatersrand Beds

At Johannesburg . .	26,400	35	15,100
„ Nigel . . .	19,800	26	8,700
„ Venterskroon . .	12,000	80	11,800
„ Krugersdorp . .	9,000	25	3,800
„ Klerksdorp . .	disturbed		

The Amygdaloidal Diabase

	Width of Outcrop. Feet.
At Johannesburg	19,400
„ Nigel	5,000
„ Venterskroon	18,600
„ Krugersdorp	nil
„ Klerksdorp	6,000

The Dolomitic Formation, including the Black Reef

	Width of Outcrop. Feet.	Average Dip. Degrees.	Calculated Thickness. Feet.
At Johannesburg	11,450	20	3,920
„ Venterskroon	5,600	15	1,450

The Gatsrand Series

This series occupies the axis of the syncline, and in the section from Johannesburg to Vereeniging it is 76,750 feet across superficially; this great thickness is, however, due to continued duplication of the beds, and probably due to the inclusion of interbedded intrusions.

SOUTHERN HEIDELBERG SYNCLINE

The Witwatersrand Beds

	Width of Outcrop. Feet.	Average Dip. Degrees.	Calculated Thickness. Feet.
At Rietbult	2,500	60	4,300
„ Heidelberg			
„ Roodepoort mine	3,000	33	1,600

The Amygdaloidal Diabase

This occupies the whole area, inside of the syncline, with the exception of a portion towards the south-eastern end which is covered by coal measures. It extends from the banket beds on one side to those on the other side, an average distance of 24,000 feet.

The Dolomite and Black Reef Formations

These do not occur in this syncline.

The Rietkuil Syncline.—This is an area about 10 miles long in a direction N.N.E. and S.S.W. and about 4 miles broad, situated about 10 miles west of Klerksdorp, as shown in the map facing page 8.

Its surface is occupied principally by Witwatersrand Beds in which various banket beds outcrop, the most important of which are known as the Elandslaagte, Africander, and Oceana Reefs.

This portion of the Witwatersrand Beds is separated from that portion which lies conformably with the main syncline by a considerable extent of igneous rock, and other intrusions have caused these beds to be so bent that along one section they appear bent into a synclinal fold.

Age of the Witwatersrand Beds.—The superficial extent of these beds in the Southern Transvaal and Orange River Colony has been shown in these times to be limited to one large and one small syncline and to the particular occurrence of the Rietkuil Syncline. There are evidences which show that denudation and disruption are responsible for these separate occurrences, all of which were doubtless at one time parts of an unbroken and continuous whole, such as was not in any way limited, as is the present extent of these beds.

The distribution in South Africa of rock similar to the banket of the Witwatersrand Beds is large. Struben represents outcrops of banket at various places south of the Zambesi, but the principal occurrences of it outside of the Transvaal Colony are on the Table Mountain of Cape Colony and in Zululand. The conglomerate of these places is auriferous.

Geologists are not entirely agreed as to the geological age of this series.

The following are two tables :—

(1) Compiled by Walcot Gibson, showing the succession of formations in South Africa according to the various students of the subject.

(2) Showing the succession as interpreted by Draper.

No. 1 [1]

Correlation of Strata in South Africa

	Schenck.	Penning.	Dunn.	Molengraff.	De Launay.	Gibson.	
14	Stormberg beds	High Veld beds	Coal measures	Coal-bearing formation	Karoo	Karoo	14
13	...	Kimberley beds	Upper Karoo	...	...	Dwyka conglomerate	13
12	...	...	Kimberley shale	...	...	...	12
			(Unconformity)				
11	Kap formation	Klip River series	Lydenburg beds	Gatsrand formation	Gatsrand formation, carboniferous Draper	Zwaartebergen quartzites	11
10	...	Dolerite series	...	Malmani formation	Magaliesberg beds and dolomite	Bokkeveld beds	10
9	...	Witwatersrand beds	...	(Unconformity)	...	...	9
8	...	...	...	Boschrand beds	(Unconformity)	Table Mountain sandstone	8
7	...	...	...	Diabase	Black Reef series	...	7
6	...	...	...	Witwatersrand beds	Klipriversberg amygdaloid	Witwatersrand beds (age very uncertain)	6
5	...	...	...	...	Bokkeveld beds (wanting in Transvaal Colony)	...	5
4	...	...	...	...	Witwatersrand beds—Table Mountain sandstone	...	4
3	...	...	...	...	Quartzite and shales	...	3
			(Unconformity)				
2	Swazi schists	Kaap Valley beds	Namaqualand schists	Older schist formation	Malmesbury beds	Malmesbury schists	2
			(Unconformity)				
1	...	Schists and gneiss	Schists and gneiss	Gneiss, schist, and granite	Granite and gneiss	Metamorphic rocks	1

[1] Walcot Gibson, Federated Institute Mining and Mechanical Engineers, vol. 12.

No. 2[1]

Table showing the Succession of the Primary Systems in South Africa, and their Correlation with European Systems

South African Primary Systems.	Where occurring in the Cape Colony and Neighbouring States.	Where occurring in the Transvaal Colony.	European Equivalents.
Quartzites	Zwarteberg, Zuurberg, Witteberg, Grahamstown	Magaliesberg, Gatsrand, Zwart Ruggens, Lydenburg ranges	Middle Carboniferous.
Slight Unconformity.			
Dolomitic Limestone or Elephant Rock	Han-Ami Plateau, Damaraland, Kaap Range, Griqualand West, Cango? Cape Colony	Malmani, Zeerust, Lichtenburg, Klerksdorp, Heidelberg, Potchefstroom, Lydenburg, Zoutpansberg, and Waterberg	Lower Carboniferous Mountain Limestone.
Unconformity.			
Bokkeveld beds	Bokkeveld, Ceres, and Southern Coast Ranges	Wanting	Upper Devonian.
Table Mountain Sandstone or Conglomerate Series	Table Mountain, Cape Town, and other hills and mountain ranges on the South Coast, Table Mountain, Natal, Zululand, Swazieland	Witwatersrand, with local conformity to underlying series, Zuikerboschrand, Rooi Kopjies, Heidelberg, Klerksdorp, Lydenburg, Waterberg, and other parts of Northern Transvaal	Lower Devonian Old Red Sandstone.
Great Unconformity generally.			
Lydenburg beds (Dunn) Swazi beds (Schenck)	Wanting	Lydenburg, Swazieland, De Kaap, Letaba, and other places in the Northern Transvaal	Silurian or Cambrian and Archæan.
Malmesbury beds (Dunn) Namaqualand schists (Schenck)	Malmesbury and Namaqualand, Natal, Zululand	Witwatersrand lowest beds and the Northern District of Transvaal Colony	

Basement Rocks—Gneiss, Intrusive Granite, Granite.

[1] By David Draper, "The Primary Systems of South Africa," read before the Geological Society, South Africa, 13th May 1895.

With respect to its geological age, the broadest statement of its position is that of Walcot Gibson, who says, in a paper read before the Federated Institute of Mining and Mechanical Engineers, vol. 12: "The age of the banket is newer than the Metamorphic series, but older than the upper portion of the Karoo, to any portion of the geological scale between these limits it can be assigned. . . . In Cape Colony gold is found in deposits of Karoo age; and as the Karoo deposit also contains bands of conglomerate indistinguishable from those of the Rand, the age of the auriferous conglomerates cannot be settled by petrological or mineralogical data."

Draper, in the above table, gives us a statement of the position of the Witwatersrand Beds reduced to within more narrow limits by correlating it with the Table Mountain Series and with the Lower Devonian or Old Red Sandstone in the European sequence. Andrew Wylie and Rubig in 1859, and later E. J. Dunn and Otterkar Friesmantel, correlated the Table Mountain Series with the Lower Devonian; whilst Sawyer and Schenck regard the Table Mountain Series and the Witwatersrand Beds as being of the same age and contemporaneous with the Lower Devonian or Old Red Sandstone of Europe.

Sawyer, Draper, and Schenck consider that the Table Mountain Series extended over an area of South Africa to include all the now existing detached portions of banket, and that it is from 8000 to 10,000 feet thick.

Walcot Gibson finds it impossible yet to agree with this, as the ages of the various detached pieces of banket seem to be different. For instance, in the Transvaal Colony there is marked unconformity between the Karoo and Banket formations, whereas Struben in a section through Zululand shows conformity between the Karoo

and a banket formation, as though they were of the same age.

The consensus of opinion would thus appear to be that the Witwatersrand Beds may be correlated with the Table Mountain Sandstone of Cape Colony, and with the Lower Devonian of Europe.

CHAPTER III

BANKET BEDS AND BANKET

It has been pointed out in the previous chapter that auriferous banket occurs, chiefly in the Witwatersrand Beds, and to some extent in the underlying Quartzite-Shale group, whilst in overlying formations it is represented by a single bed—the Black Reef. Their general geological relationships have already been dealt with, but it now remains to describe these various occurrences in greater detail in their stratigraphical sequence, commencing from the lowest.

Banket in the Quartzite-Shale Group

This group has suffered considerable deformation; it lies on the granite, and those forces which so altered that rock that gneiss and schist resulted, have also altered this overlying group, so that in places the origin of some of its component beds is in question.

In consequence of this deformation, the beds are generally distorted and broken, and the regularity found in the overlying strata does not characterise the group now under consideration. As stated in describing this group, auriferous banket beds occur in it at various points around both synclines.

The most important development of this banket has been made at the New Rietfontein Estate and Rietfontein A, two properties situated about 9 miles E.N.E. from Johannesburg and about 2½ miles north of the line of the Main Reef Series.

In these mines there are three principal reefs, known respectively as the North Reef, Middle Reef, and the Leader, and collectively as the Du Preez Series.

The North Reef varies from 8 inches to 36 inches in thickness; it lies upon slate, about 30 feet north of the Middle Reef; its pebbles are small and its gold content irregular. The Middle Reef and the Leader generally occur in such close proximity that they are mined together; the former averages about 10 inches in thickness and the latter is smaller: they both consist of small-pebbled banket with a considerable amount of black sulphides, whilst, in certain zones, which are frequently poorer, rounded pyrites often occurs. The gold content of these two reefs is generally high, though very irregular; in places this irregularity is so pronounced that it is found impossible to obtain any reliable estimate of the ore reserves by sampling; visible gold is frequently seen both in the oxidised and unoxidised ore. In No. 2 shaft of the Rietfontein A a pronounced ore shoot has been met with; it extended for several hundred feet, the greater portion being east of the shaft.

In addition to these reefs, there is another called the Stable Reef; the outcrop of this reef is about 200 feet south of that of the Middle Reef. The alteration which the banket of these reefs has undergone is shown by the nature of the cement, which has a burnt appearance and is very chloritic.

It is considered that this same series of reefs has been struck in the Quartzite-Shale group near the township of Florida, about 7 miles west of Johannesburg.

In the property of the Alexandra Estate, situated some 3 miles north of Krugersdorp, banket reefs have been opened up in prospecting operations; one of these is known as the Alexandra Estate Reef, and of this one Mr. R. J. Frecheville says in a report "that the reef is in mode and occurrence like the Du Preez Reef, Rietfontein Estate." It is small and of irregular value; in the footwall, the quartzite, for 12 or 18

inches, often carries gold, and frequently the quartz veins which cross the reef show visible gold.

There are evidences which would show that the reef worked in the Molyneux mine, about 8 miles S.E. of Heidelberg, is also to be classed in the Quartzite-Shale group. This reef occurs at a contact of quartzite and "shale," this latter being the footwall; in places it has been found to leave this position and to extend into the quartzite; in size it varies from 1 to 36 inches, and in value from a few dwts. to several ounces of fine gold per ton.

At Rietbult, on the north-eastern side of the Heidelberg Syncline, a reef, shown in position in the section, Fig. 12, also occurs in this group of quartzites and shales: it was, however, only auriferous to the extent of a few dwts. per ton. Between Krugersdorp and Klerksdorp, as previously stated, several banket beds occur in this group, but so far no extensive development has been done on them.

Banket in the Witwatersrand Beds

In describing the occurrence of banket in these beds, the same natural division of the Witwatersrand Syncline into districts which was used before, is introduced again.

The complete series of banket beds in this formation is generally considered to be the following, placed in order of sequence, commencing with the lowest:—

Main Reef Series.
Livingstone Reef Series.
Bird Reef Series.
Kimberley Series.
Elsburg Series.

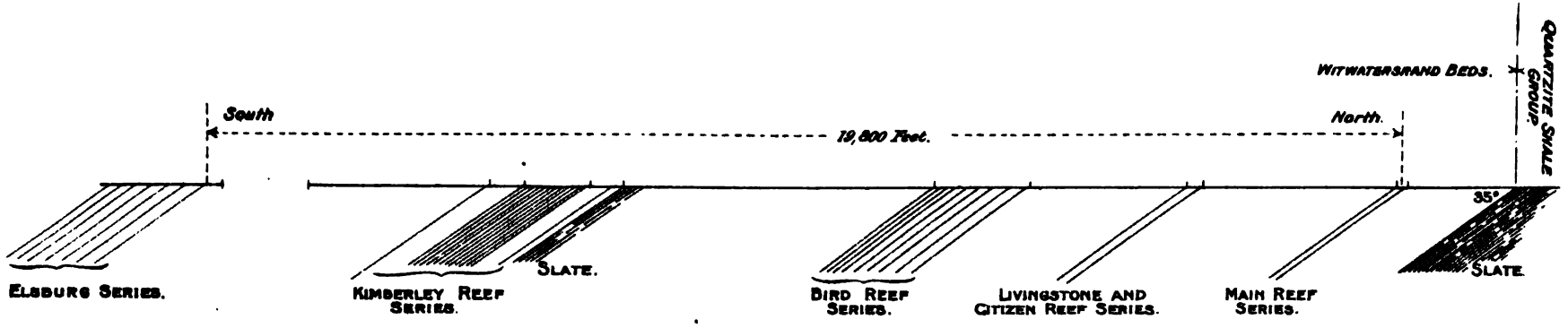
Central District.—In this district the relative positions of the different series mentioned above is shown in the general section, Fig. 32, and in the two sections, Figs. 1 and 2, one of which is taken across the formation at Johannesburg, and the other near Roodepoort.

In the former, Fig. 1, it is probable that the distances between the first four series, starting from the Main Reef, are approximately correct, because that between the Bird and Main Reef Series was determined from the occurrence of these two reefs in the Robinson Deep shafts, where the section afforded was, as far as could be judged from a local knowledge of the dislocations, free from any material disturbance; similarly, that between the Bird and Kimberley Series was determined along a line on surface, where, after a detailed examination, it appeared that no large dykes were likely to cross; the position of the Livingstone Series was also determined along a section which promised equal freedom from disturbance. With regard to the position of the Elsburg Series, an average has been struck between the distances which have been given by various authorities.

The State Mining Engineer in his section from the Pyramids to Vereeniging, included in his report for 1896, places this series 26,690 feet south of the Main Reef; Messrs. Hatch and Chalmers in their book, *The Gold Mines of the Rand*, show it as extending from 17,160 feet to 20,060 feet south of the Main Reef Series; and M. de Launay gives the outside limit of the thickness of strata between the two series as 6485 feet, which, at an average angle of 35°, gives about 11,200 feet on surface; this last estimate, however, is a general one and not confined to any one section.

The section near Roodepoort, Fig. 2, is taken from the Vogelstruis Estate to the Great Britain, and represents the distances as measured on the surface; the Bird Reef is a well-developed body of banket at this place, though for lack of information its position has only been roughly indicated.

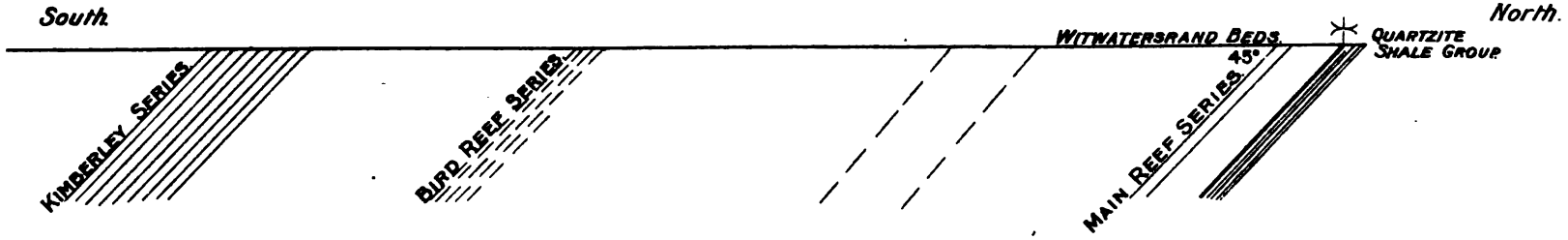
Main Reef Series.—This is the first banket series met with overlying the Quartzite-Shale group; it is separated from the shales of that group by an extent of quartzite which has the following thicknesses: at Roodepoort some 200 feet, at Johannesburg some 500 feet, and at Boksburg about 125 feet.



Section across Formation at Johannesburg.
Projected at an average dip of 35°

Scale 1333 ft. - 1 inch.

Fig. 1.



Section across the Formation from the Vogelstruis Estate to the Great Britain.
Compiled from surface data.

Scale 1,000 feet - 1 Inch.

Fig. 2.

To face Page 44

It is mainly from this series that the gold is being obtained; it consists principally of three reefs, which, commencing from the lowest, are called respectively—

The Main Reef.
The Main Reef Leader.
The South Reef.

In addition to these, there are many other less important banket beds which are roughly classified as under:

The North Reef, including all the beds north of the Main Reef.

The Middle Reef, including all the beds between the Main Reef Leader and the South Reef.

The Bastard South Reefs, including all the small-pebbled banket beds immediately overlying the South Reef.

A detailed consideration of the Main Reef Series, the importance of which causes it to merit special treatment, will be found in the following chapter.

Livingstone Series.—This series has not received much attention in this district, as it has always proved to be poor. It consists of a limited extent of banket, the pebbles of which are so small that it passes imperceptibly into a grit.

Its outcrop on the Crown Deep lies 2000 feet south of the Main Reef, and it there consists, principally, of two beds respectively 12 inches and 48 inches in thickness, about 250 feet apart on surface.

On the Village Main Reef this series is about 1400 feet south of the Main Reef.

A bore-hole put down just outside the southern boundary of the Cinderella, about one mile east of Boksburg, completely sectioned this series; at that point it was found to lie about 1000 feet south of the Main Reef, and to consist of eleven layers of small-pebbled banket, with a total thickness of 75 inches, interstratified throughout some 150 feet, true thickness, of quartzite, and giving an average assay of about 6 grains per ton, one piece 12 inches thick assaying 2 dwts.

It is this series which, farther to the east in the Van Ryn-Chimes district, is known as the Chimes Series; this has proved to be in places payably auriferous.

Bird Reef Series.—This series consists of several beds of small-pebbled banket; the pebbles are principally white, though some black and yellow ones occur, and in its overlying beds some red ones are found. The different beds of this series south of Johannesburg are shown in the section, Fig. 3, which was obtained from a bore-hole put down to the south of the Wemmer.

In this bore-hole the aggregate true thickness of banket was 76.9 feet, and that of the interstratified quartzite 322.3

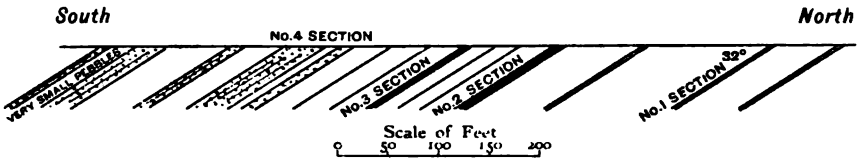


FIG. 3.—Section of the Bird Reef Series as projected at a dip of 32°; from the results of a bore-hole put down south of the Wemmer.

feet, so that this series covered a true thickness of 400 feet; the average assay of the banket was but a few grains per ton, and the highest assay obtained was not much more.

On the property of the Vogelstruis Consolidated Deep, one of the reefs of this series, 3 feet to 4 feet thick, has given prospects of from 5 to 7 dwts. per ton.

On the Bantjes Consolidated, the Bird Reef outcrop is 3800 feet south of the Main Reef; it there consists of several parallel beds, on one of which some prospecting work has been done; this one is 4 feet to 5 feet thick, and it is stated to assay from 4 to 7 dwts. to the ton on an average, whilst some very much higher assays are at times obtained.

South of the Consolidated Main Reef property, a considerable amount of work has been done on this series; the ore obtained was for a time treated by direct cyaniding.

At a place south of the Langlaagte Royal, a good deal of

ore was stoped out in one place from this reef; from that point to the Crown Reef mill-site, on the east side, and to an undetermined extent on the west side, there is a duplication of this series, forming two outcrops on surface, caused by a large longitudinal dyke.

All along this portion of the Rand this series is divided into four principal portions numbered 1, 2, 3, and 4, from the lowest up, as in Figs. 3 and 4; of these, No. 1 has the largest pebble, and on it most of the prospecting work has been done; No. 2 consists of a series of grits; No. 3 is a bed of banket of which the cement is so strong that the reef outcrops very markedly; and No. 4 consists of a large width of grits and

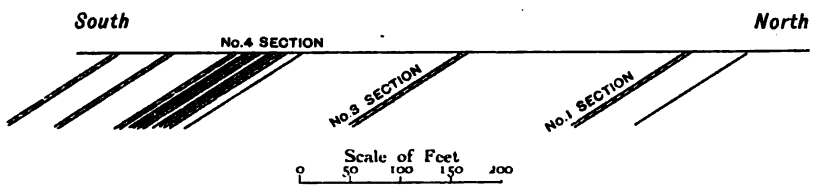


FIG. 4.—Section of the Bird Reef Series as obtained on surface near the Crown Reef Battery.

small-pebbled banket, none of which have ever proved to be more than very slightly auriferous.

This series has been struck in both shafts of the Robinson Deep: in the West shaft only partially, and in the East almost completely; in these places the banket rarely assayed more than a few grains per ton.

East of this mine its outcrop is again exposed south of the Henry Nourse, and from there it can be followed almost continuously eastwards, till it becomes hidden beyond Boksburg; it crosses the spruit just north of the railway embankment at Elandsfontein; it has been met with in the two shafts of the Knights Central, where it was valueless, and it occurs near the Boksburg railway station, at a distance of about 2000 feet south of the Blue Sky Reef.

Kimberley Series.—This is an extensive series of banket beds which runs from one end of the district under review to

the other; its outcrop lies at the following distances south of the Main Reef:—

At Roodepoort	. .	5300 feet.
At Johannesburg	. .	9000 ,,
At Boksburg	. .	5000 ,,

At Roodepoort it has received its greatest development, and here it consists of about thirteen distinct beds with an aggregate true thickness of about 140 feet, interstratified over 5000 feet of quartzite, as shown in Fig. 5.

The more important of these beds are known by different names. The Kimberley Reef itself is the upper and very large one; some of its pebbles are large, but generally they

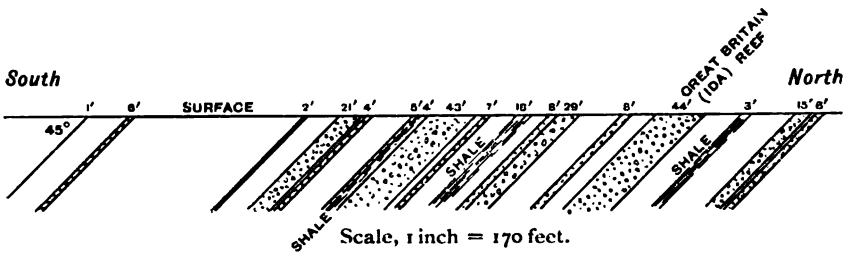


FIG. 5.—Section through the Kimberley Series of the Great Britain mine; the horizontal widths of the banket and shale beds are given in feet.

are of a good medium size. This reef is very pyritic, some of the pyrites occurring crystallised, either disseminated throughout the cement or in veins, and some as rolled pebbles. Owing to oxidation, this reef on the surface has a yellow or red appearance. The reef, which overlies the Kimberley Reef, is characterised by its white appearance on the surface and the comparative absence of pyrites. The other reefs have not been prospected much, as they have been found to be poor.

In the Marie Louise, other reefs of this series are worked, in which zinc blende and copper pyrites have been found in addition to the ordinary pyrites. The pebbles are generally of white and black quartz, though pebbles of igneous rock have been found.

No occurrence of shale is known with this series at the Great Britain mine. There is, however, a dyke which follows along the formation just north of the Kimberley Reef, and which at its outcrop has become weathered to look like shale, and there is on either side of the reef a bed of soft micaceous and finely-laminated quartzite.

South of Johannesburg there is an occurrence of calcareous shales in the footwall of this series, as indicated in Fig. 1. Near Boksburg this series has been opened up by the Leewpoort and Ziervogel companies.

Elsburg Series.—This is an extensive series of banket beds, which lies at Johannesburg, about 19,800 feet south of the Main Reef, and at Elsburg about 15,000 feet south.

Messrs. Hatch and Chalmers, in their section to the Black Reef, show this series as interstratified over a width of 2900 feet on the surface, whilst M. de Launay places the aggregate true thickness of the banket beds as 65 feet.

On Rass's farm south of Johannesburg a considerable amount of prospecting has been done on this series, which is there very extensive, continuing right up to the overlying diabase. The pebbles in places are very large, sometimes being several inches in diameter, and they are principally of white quartz or light-coloured quartzite.

On the farm Elandsfontein, about 6 miles south-east of Johannesburg, some prospecting work on this series is now being done, and there it would not appear to be so extensive. There are two beds outcropping, and underground a third has been discovered. These all occur within a vertical distance of about 290 feet. The underlying one has so far been found to be the best, though its value is not above three or four dwts. to the ton. It is, however, improbable that these represent more than a portion of the whole extent of this series.

The outcrop of this series westward of Johannesburg is not so well marked.

Van Ryn-Chimes District.—This district is taken to extend from the Apex mine to Klipfontein. The succession of beds in this district is shown in the section in Fig. 6, which, from the Bird Reef to the Main Reef, has been obtained from the Chimes mines bore-hole.

There are four principal banket beds called respectively, and commencing with the lowest—

(1) The Main Reef Series.
(2) The Chimes Series.
(3) The Bird Reef Series.
(4) The Kimberley Series.

In addition, there is the Elsburg Series, but so far the presence of this has only been demonstrated by analogy, as at its horizon the surface is covered by more recent coal measures.

Of these different series, numbers 1, 3, and 4 have been correlated with those which occur similarly named in the Central district. The only series about the correlation of which there is any question is the Chimes Series.

Main Reef Series.—This series is represented principally by two banket beds, separated by about 10 feet of quartzite. Of these the lower and larger one is called the Main Reef, and the upper and less persistent one the Main Reef Leader. As demonstrated in the following chapter, these two reefs are respectively the actual extension in this district of the two which are similarly named in the Central District. The South Reef proper is barely represented here. Its deterioration going eastwards from the Central Rand can be gradually followed till, in the East Rand mines near Boksburg, it is practically a bastard reef, and in the Modderfontein property there is no evidence to show that it any longer exists.

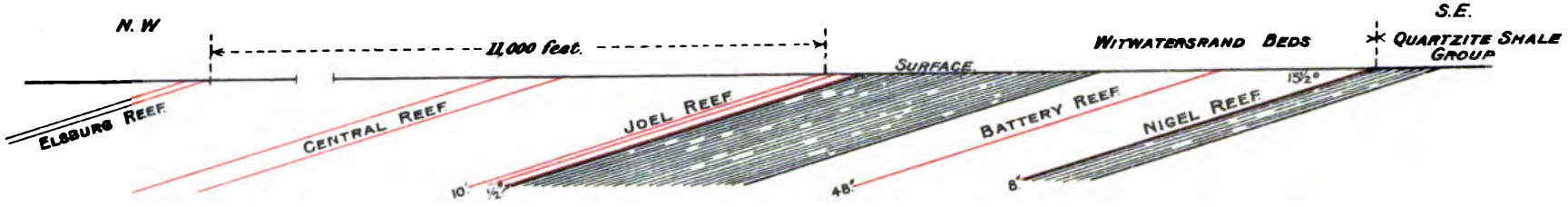
Chimes Series.—This series has been exploited to the greatest extent in the Chimes mine about 6 miles E.N.E. of Boksburg. In this mine it consists of four principal beds of banket called respectively, commencing from the lowest,

To Face Page 50

Section across the Formation principally projected from the Chimes Mines Borehole.

Scale 1,000 ft. - 1 inch.

Fig. 6.



Section across the Formation at the Nigel reproduced from a Vertical Section

Scale 1000 feet - 1 Inch.

Fig. 7.

the North Reef, Bastard Reef, Chimes Reef, and the South Reef, the relative positions of which are shown in the following section (Fig. 8):—

The North Reef is characterised by the smallness of its pebbles and the scarcity of them. It is 12 inches thick and of low grade.

The Bastard Reef is small, and contains practically no gold.

The Chimes Reef consists of a narrow but rich leader from 3 to 4 inches thick, separated by an average distance of about 2 feet from a larger body 16 to 24 inches thick. The leader carries nearly all the gold.

The South Reef was worked to some extent on the sur-

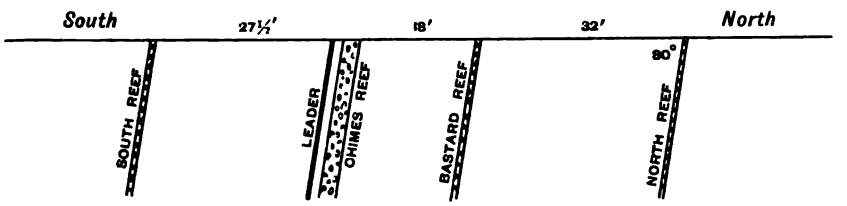


FIG. 8.—Section of the Chimes Series in the Chimes mine. Scale, 30 feet = 1 inch.

face, but it has hardly been touched since. To the east and west, and beyond the immediate neighbourhood of this mine, this series gradually loses its economic value through deterioration in gold content.

The particular reefs which are met with in the Chimes mine are not defined in the neighbouring mines working the same series, as shown on the subjoined average section (Fig. 9) across the formation in the property of the Van Ryn West, in which the relative positions of the various reefs of the Main Reef Series and the Chimes Series are indicated. Of the latter the South Reef alone is of any value.

The general features of this series are as follows:—The reefs are small beds of banket, which pinch out and are quickly replaced by others. They are contained in quartzite, which also carries gold, and throughout which, within the limits of the horizon of this series, pebbles are scattered.

The banket itself consists of small black and white pebbles of quartz, with some imperfectly rounded slate pebbles. There are no large pebbles such as are carried in the Main Reef Series.

These features are identical with that series of small-pebbled banket beds which were struck in a bore-hole to the south of the Cinderella, and with that series these reefs must be classified as being the eastern extension of the Livingstone Series of the Central district.

Bird Reef Series.—The occurrence of this series in this district is very well represented by the section obtained in

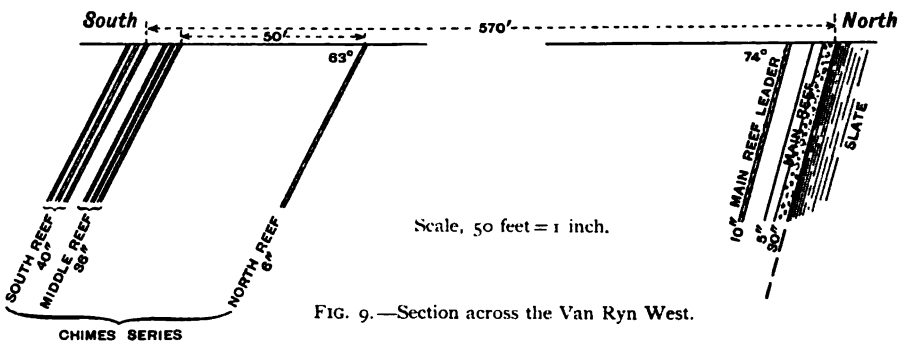


FIG. 9.—Section across the Van Ryn West.

the Chimes Mines bore-hole, details of which are given below. It persistently maintains the characteristics which mark its occurrence in the Central district, and so far has proved to be of no economic value.

Kimberley Series.—This series outcrops markedly in this district. Its position in the section, Fig. 6, is that given by M. de Launay. It consists of an extensive series of banket beds which are only slightly auriferous, and of which the pebbles are generally very large.

Elsburg Series.—As stated before, this series does not outcrop in this district, as its horizon is covered with coal measures.

The following are some particulars of the banket reefs which were struck in the bore-hole put down on the Chimes

Mines property at a distance of about 4340 feet from the outcrop :—

Bird Reef Series.—From 210 to 228 feet there were six beds of small-pebbled banket, which gave an aggregate thickness of 66½ inches, and had an average assay value of 11 grs. per ton.

From 430 to 454 feet there were four beds of similar banket of an aggregate thickness of 47 inches, and an average assay value of 12 grs. per ton.

Chimes Series.—From 1154 to 1172 feet there were five beds of small-pebbled banket with an aggregate thickness of 32 inches, and an average assay value of about 5 grs. per ton.

Main Reef Series.—The following beds belonging to this series were struck :—

No. 1	at 1540	feet 10	inches,	4½	inches thick,	assayed	traces.		
„ 2	„ 1738	„ 0	„	4	„	„	14	dwts. per ton.	
„ 3	„ 1754	„ 0	„	36	„	„	16	„	

The country rock was quartzite, in which several small beds of shale were interbedded.

At a depth of 1788 feet the drill passed into a strong shale formation, which continued to a depth of 2500 feet, when quartzite was again encountered. In this extent of shale two sheets or dykes of igneous rock were encountered.

At a depth of 2884 feet the quartzite gave place to igneous rock, in which the drill was still boring at the bottom of the hole, a depth of 3000 feet.

Assuming the average dip to be 30°, this succession of reefs and strata is represented in the section, Fig. 6.

Nigel and Heidelberg District.—This district extends from the farm Vlakfontein, about 5 miles N.E. of Nigel, down the Blesbok Spruit, to a point about 17 miles S.S.W. from Heidelberg.

Along this whole length the overlying banket beds,

especially those which are correlated as the Kimberley Series, can be traced to extend, whilst the underlying banket beds are for some distances concealed under igneous overflow.

The succession of banket reefs in this district, commencing with the lowest, are as under :—

(1) The Nigel Reef (Main Reef Series).

(2) The Battery Reef (Bird Reef Series).

(3) The Joel Reef and the Central Reef (Kimberley Series).

(4) The Elsburg Reef (Elsburg Series).

The relative positions of these are shown in the section, Fig. 7, which is taken across the formation at the Nigel, and which, from the Nigel Reef to the Central Reef, was obtained from a bore-hole, the position of the Elsburg Series being from surface measurements.

Nigel Reef.—The characteristic occurrence of the Nigel Reef is that of a small band of banket lying at the contact of overlying quartzite and underlying shale; it is "frozen" into the former and separated from the latter by a thin layer of clay, often accompanied by secondary vein quartz; overlying it, and separated from it by a foot or so of quartzite, there is often a "bastard" reef composed of small pebbles. The characteristics of the reef itself are a small thickness with a large range of assay value, and the occasional occurrence, amongst the ordinary black and white pebbles, of vari-coloured pebbles.

When the thickness varies it is generally found that in any limited area the assay value varies inversely with the thickness.

These characteristics are practically identical with the Main Reef on Modderfontein and Klipfontein of the Van Ryn-Chimes district, so that this reef is likely to be the actual extension in the Nigel district of the Main Reef of the Central Rand.

The Battery Reef.—This is a banket reef about 4 or 5

feet thick, of which the pebbles are white and black and of a small size; neglecting the overlying bed of shale, shown in the section, Fig. 7, its position between the Nigel and Joel Reefs, which are respectively classed as the Main Reef and Kimberley Series, would point to a correlation with the Bird Reef Series, which is confirmed by the similarity of the banket. This being the case, the Livingstone Series is not noticeably represented.

The Joel Reef.—This reef is a large reef which can be traced throughout this district. It is often spoken of as the "striped pebble reef," because it is characterised in several places by the occurrence of banded black quartz pebbles. This reef is sometimes compact. In one bore-hole it consisted of 9½ feet of solid reef with no outlying layers, but it is more often broken up into several beds. At its outcrop on the property of the Sub-Nigel Ltd., it consisted of several beds with an aggregate thickness of 7 feet of banket, interstratified with 5½ feet of quartzite; the pebbles were of medium size, except in the footwall, where large pebbles often occurred. This footwall portion carried gold varying in amount from traces up to 40 dwts. per ton, but the other parts were poor.

In No. 2 shaft of the Central Nigel Deep this reef was struck at a depth of 596 feet, and at 633 feet it was still in the bottom of the shaft; it consisted of several banket beds varying in thickness from 6 to 30 inches, and alternating with quartzite.

As seen from the section, this reef overlies a large bed of slate. At the contact with this there is a layer of banket one pebble thick, which only assays traces of gold, but which is persistent over long distances.

Central Reef.—This reef is so named because it was struck in the shafts of the Central Nigel Deep; in No. 2 shaft, at a depth of 140 feet, it was 6 feet thick, of rather large pebbles and assayed traces; in a bore-hole close by, it was dispersed over a much greater thickness and the pebbles were smaller.

This reef and the Joel Reef have together characteristics of occurrence and petrological features which make it appear probable that they are rightly correlated with the Kimberley Series.

The Elsburg Reef.—This reef lies at some distance overlying the others. Its position is similar to that which the Elsburg Series occupies in the Central district, but it is not such an extensive reef series. It can, however, be traced as running parallel with the Joel Reef throughout the Nigel and Heidelberg district.

Venterskroon District.—The following section across the banket beds in this district is obtained from a complete section

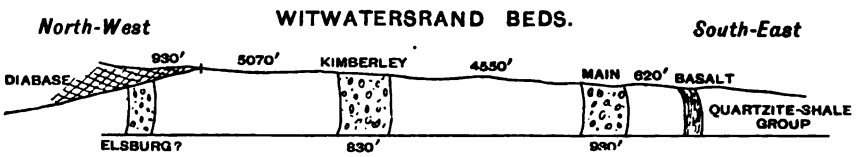


FIG. 10.—Section across the outcrop of the Witwatersrand Beds in the Venterskroon district.

across the formation from the farm Leewfontein on the north-west to Koppieskraal on the south-east, the line of section lying some 5 miles north-east of Venterskroon. This complete section was given by Mr. H. B. Bunkell in his paper "The Venterskroon Goldfields," Fed. Inst. Mining Engineers, 24th December 1896, and from this paper the following tabulated statement of the reefs, with their probable correlations, was also principally obtained :—

Name of Reef.	Probable Correlation.
Acme or Odin Red or Meisters North or Jumbo Unnamed	Main Reef Series
Unnamed Yellow or Ross Myrtle Other thin beds	Bird Reef Series

Name of Reef.	Probable Correlation.
Stink Reef	Kimberley Series
Gordon Black Seams	
Two Foot Reef	
Springbok or Brown	
Great Western Reef	
Leach or Amazon	
Other leaders	

According to this statement the Elsburg Series is not represented on the surface; its probable position under the diabase is indicated in the section.

The correlation of those reefs, which lie nearest the Quartzite-Shale group, as the Main Reef Series, places that series in its proper horizon.

On the property of the Rooderand G. M. Co. the Springbok Reef is a large body varying from 30 to 40 feet in thickness, and dipping at an angle of 43° to the south-east. One section of this large reef which has been opened up is of better grade than the bulk, averaging for certain lengths from 4 to 10 dwts. to the ton, over a width of 5 feet. The Great Western Reef varies from 7 to 12 feet in thickness, and over a portion, which averages 4 feet in thickness, it gives much the same assays as does the Springbok Reef.

The Odin Reef is smaller, and better assays have been obtained from it.

Klerksdorp District.—In this district the continuity of the reefs, with the exception of the Black Reef, is very much disturbed by dyke intrusions and covered up by dyke overflows. It is because of this that the relative positions of the reefs here afford, in the absence of a detailed geological survey, very little data on which to base a correlation, so that their petrological features have to be more particularly studied. In the first place there is in this district a reef which in its occurrence is quite distinct from the ordinary run of banket beds. This is the Buffelsdoorn or White Reef. In this the gold is associated with an occurrence of bituminous matter in grayish

quartzite and with the occurrence of igneous rock in the hanging wall. The bitumen chiefly runs as threads along cracks, though to a small extent it also occurs throughout the quartzite.

Mr. Denny, in his book *The Klerksdorp Goldfields*, p. 88, states that "the ore without coal contains a very large percentage of fine iron pyrites, and is as a rule practically valueless"; and on p. 89, "It has been pretty conclusively proved that the gold zone in the White Reef is confined within certain distance of the dyke, maintaining throughout a well-defined parallelism, and beyond this limit no payable rock occurs."

This being the case, it is implied that the igneous rock and bitumen are jointly responsible for this occurrence of gold, and as it is reasonable to suppose that the latter could have only effected the deposition, the responsibility for the presence of the gold rests with the igneous rock. This rock is either a portion of the diabase which underlies the Black Reef, or it is a separate intrusion lying closely under that diabase. The other reefs in the district may be roughly classified into two groups:—

(1) Large-pebbled reefs and reefs associated with them.

(2) Small-pebbled reefs.

Large-Pebbled Reefs.—Amongst these are included the Gold Estate Reef, the Commonage Reef, the Ada May Reef, and the reefs of the Rietkuil Syncline.

The Gold Estate Reef varies from 4 to 20 feet in thickness. Its pebbles are generally large, though they vary in size, the largest ones being found in the footwall and the smallest ones near the hanging wall. This reef is generally very variable in gold content, one portion of its thickness, which is not always the same portion, being usually of better grade than the remainder, though even this has not a high average value. This reef is being worked quite close to Klerksdorp, and diamonds have been found in it.

The Commonage Reef is about 5 feet thick. The size of its pebbles and the character of the cement vary greatly. It is often found that where the pebbles are above the ordinary size the grade is better. Nodular pyrites occurs frequently, and bituminous seams occasionally. This reef has not a high average assay value. It has been opened up quite close to Klerksdorp.

The Ada May Reef consists of two reefs, separated by about 50 feet of quartzite. They are parallel and in series with the Commonage Reef. The overlying one is about 12 inches thick, with small pebbles and of low grade. The underlying one is about 4 feet thick, with large pebbles, many of a very black quartz, and rounded pebbles of pyrites; it has an irregular and uncertain assay value.

There are four principal reefs in the Rietkuil Syncline, named respectively, commencing from the lowest, the Big Pebble Reef, the Africander Reef, the Worcester Hope Reef, and the Greens Reef. The Big Pebble Reef maintains a regular thickness of from 27 feet to 34 feet. Its pebbles are mostly large and of dark-coloured quartz, though some of banded and flesh-coloured quartz also occur. In addition, pebbles of igneous rock have been found.

Iron pyrites occurs chiefly in rounded pebbles, but sometimes in veins. The cement is chiefly quartzose, and very little pyrites is disseminated through it. The gold content is generally very low, but irregular.

The Africander Reef is 6½ feet thick, the Worcester Hope Reef 3½ feet, and the Greens Reef 5½ feet. These reefs are of medium-sized pebbles; pyrites occurs in the cement, and the gold is distributed regularly. In the Worcester Hope Reef some threads of bituminous matter have been found.

These reefs occur about 8 or 9 miles west of Klerksdorp, in a portion of the Witwatersrand Beds which has been detached from the main body by igneous intrusions and folded up into a syncline.

Small-Pebbled Reefs.—Amongst these are included the Elandsheuvel and Oceana Reefs. The first-named reef is situated some 3 miles north of Klerksdorp. It is about one foot thick, with a low assay value, and carries small pebbles. The Oceana Reef lies about 5 miles north-west of Klerksdorp. It is a fine-pebbled reef about 2 feet thick, and poor.

On petrological grounds the larger-pebbled reefs and those associated with them may all be considered to belong to one horizon, which is further confirmed by the concomitant occurrence of two parallel and, to some extent, interbedded dykes throughout. They possess characteristics which are identical with those of the Kimberley Series as it occurs in the Marie Louise and Great Britain mines to the south of Roodepoort. The smaller-pebbled reefs are similar to the Bird Reef Series, and as both of these reefs which have been mentioned occur lower down in the formation than the larger-pebbled beds, such a correlation as between the two types of reefs would be in regular stratigraphical order.

The Main Reef Series always lies near the junction of the Witwatersrand Beds with the Quartzite-Shale group, and this horizon in this district again underlies these small-pebbled bankets, though it is to a great extent covered up by more recent overflows of igneous rock.

On these lines it would appear that in the Klerksdorp district the Kimberley Series and the Bird Reef Series are represented respectively by the larger-pebbled and the smaller-pebbled bankets with their associated beds.

In this case the Elsburg Reef must lie under the amygdaloidal diabase, and the Main Reef Series must be hidden under more recent igneous overflows.

The Buffelsdoorn Reef is probably associated with the larger-pebbled reefs, and would thus occur at the horizon of the Kimberley Series, in which, however, it can hardly be included because of the nature of its occurrence.

Krugersdorp District.—The succession of banket series

discovered in this district in the Witwatersrand Beds is as follows, commencing from the lowest :—

The Botha's Series	corresponding to the		Main Reef Series.
The Africander Series	,,	,,	Livingstone Series.
The Monarch Series	,,	,,	Bird Reef Series.
The Battery Reef Series	,,	,,	Kimberley Series.

The relative positions of these are shown in the section, Fig. 11, which is taken across the formation about 1 mile east of Krugersdorp.

Here it may be mentioned that the distances given in this section, though they are from surface measurements, do not represent the true distances between the reefs, because in this district there is a series of faults which, striking north-east and south-west, throw down the reef on the south side. This series of faults is so marked and pronounced that any line of section across the formation must intercept one or more faults, the effect being to bring the reef outcrops nearer together than they really are. Further reference to these faults is made in the chapter dealing with the subject of faulting.

Botha's Series.—This consists chiefly of two reefs, called respectively, commencing from the lower,

The Botha's Reef,
The South Reef.

The Botha's Reef is also accompanied by several leaders. It is a large reef, the greater bulk of it being low grade. The South Reef is usually about 50 feet south of the Botha's Reef. It is a small and rich leader.

It is seen from the section that this series closely overlies a bed of shale, lying 200 feet to the north. This bed of shale extends to the southern side of the hills north of Krugersdorp, and it belongs to the Quartzite-Shale group.

The petrological character and the stratigraphy of this Botha's Series are very similar to those of the Main Reef Series of the Central Rand. The position of this Botha's

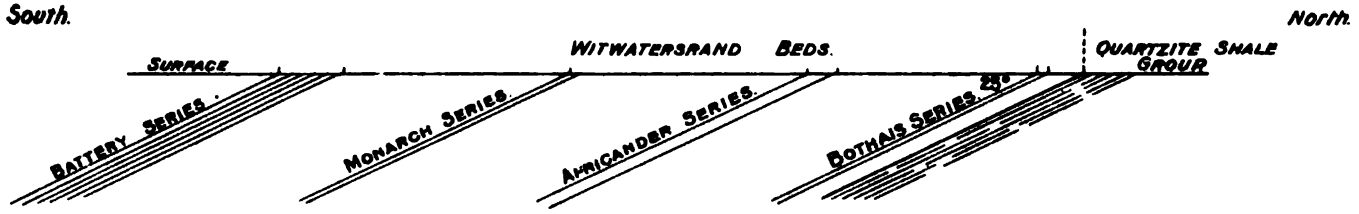
Series at the base of the Witwatersrand Beds is identical to the position held by the Main Reef Series. These two facts, strengthened by the sequence of the overlying banket series, prove that the Botha's Series is the actual extension of the Main Reef Series in the Krugersdorp district, and it will be described in detail in the chapter on the Main Reef Series.

The Africander Series.—This series consists principally of two banket reefs called the Keely and the Africander. The latter reef is about 300 feet south of the former. They both consist of small-pebbled banket, and are of no economic value. This series, in character and in its position, resembles the Livingstone Series.

The Monarch Series.—This series consists generally of three reefs: one, called the North Reef, is 2 feet wide; close to the hanging wall of this there is another, about one foot wide, called the Leader, and 100 feet to the south there is the third. The Leader has given good pannings, but the others are of low grade. This series possesses the characteristics of the Bird Reef Series, though its pebbles are somewhat larger.

Battery Reef Series.—This series consists of from ten to fifteen banket beds, which vary in thickness from a few inches up to 10 feet, and which are interstratified over a true thickness of about 150 feet. The pebbles are usually above the average size and most of the beds are poor.

Towards the footwall there is a reef about 4 feet thick, which is characterised by carrying very large pebbles, sometimes as much as 10 inches in length, and by the occurrence of rolled pyrites. These pebbles of pyrites chiefly occur in the footwall; they are often as large as a bird's egg, and, together with the crystalline pyrites, they form by weight about ten per cent of the banket bed. Visible gold is sometimes seen in the footwall with the rolled pyrites, and sometimes in cavities which have been vacated by the pyrites. At the Violet mine one pebble was found showing gold along a crack

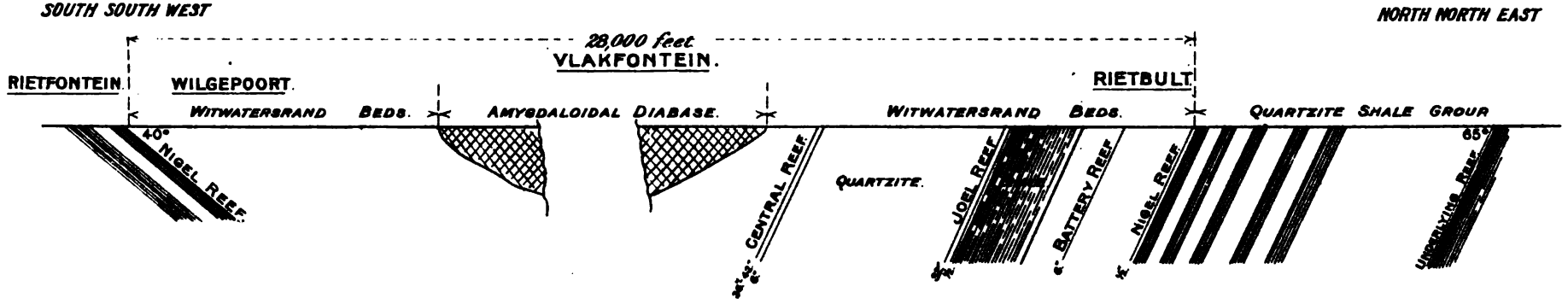


Section across the Formation about One Mile east of Krugersdorp from the Bothais Series to the Battery Series.

Scale 1000 feet = 1 inch.

Fig 11.

To face Page 62



Section across the Southern Heidelberg Syncline.

Scale 1000 feet = 1 Inch.

Fig. 12.

within its mass. In driving on the first level of this mine from the old Flora shaft, the reef at a certain point curved out into the hanging wall and then curved back across the drive into the footwall, where they continued for a short distance, after which they abruptly turned back into their original position. This excursion from regularity was accompanied by a layer of graphitic shale about 18 inches thick, which, as an outside curve, enveloped the reefs. It is stated that at this place a piece of petrified wood was obtained. The character of this series and its position are such that it must be correlated with the Kimberley Series of the Central Rand.

The Elsburg Series.—The horizon of this series in this district is covered by more recent measures, and this reef has not yet been discovered here.

Southern Heidelberg Syncline.—A section made across this syncline at its narrowest place, from Rietbult on the north-eastern side to Wilgepoort on the south-western side, is given in Fig. 12. In this section the reefs are named in conformity with those in the Nigel and Heidelberg district, because they are similar in all respects, so that the correlation which applied in that district is applicable here, as under :—

Nigel Reef corresponding to the Main Reef Series.
Battery Reef „ „ Bird Reef Series.
Joel Reef and Central Reef corresponding to the Kimberley Series.

The Elsburg Series does not outcrop. In this section also the position of a reef in the Quartzite-Shale formation is shown.

On the Wilgepoort side the Nigel Reef only is indicated, but the other reefs also occur.

The reefs in the Witwatersrand Beds, as they are found on Rietbult, are described as under.

Nigel Reef.—This reef here possesses all the characteristics of the same reef at Nigel; it occurs at the contact of

overlying quartzite and underlying slate. The banket is usually of high assay value, but its average thickness is very small, being about half an inch.

The Battery Reef.—This reef consists of fine pebbles in a hard siliceous cement; it is about 6 inches thick, and of no economic value.

The Joel Reef.—This reef is about 32 inches thick, of medium-sized pebbles, about the size of a small walnut. It closely overlies a bed of slate, as it does at the Nigel, at the contact with which there is a layer of pebbles, which is also shown in the Nigel section. It is generally of low grade.

The Central Reef.—This reef generally consists of two or more layers of banket which are only very slightly auriferous; the pebbles are medium-sized. Although in this section it is shown as lying about 1000 feet from the Joel Reef, the distance between the two is generally much less.

The Black Reef Formation.—The extension of the Black Reef formation on surface has been described. From the surface it dips at a lower angle than the underlying Witwatersrand beds, the average angle of dip being from 10° to 15°. One of the deepest points at which it has been encountered is on the property of the Midas Deep, about 12 miles south of Krugersdorp, where, in a bore-hole placed about 2000 feet from the outcrop, this reef was struck at a depth of 795 feet; it was there 13½ inches thick, and was stated to assay 88 dwts. of fine gold per ton.

Though the formation can be followed right around the Witwatersrand Syncline, the reef itself is not always present; it has been opened up only along the north-west confine of the syncline, from Klerksdorp through Krugersdorp to the Black Reef district, which lies about 9 miles to the south-east of Johannesburg, and includes those mines which are grouped around the Orion mine.

Klerksdorp District.—The Black Reef in this district has been opened up principally in the Eastleigh group of mines.

The following description of this reef is taken from Appendix E, by G. Kubale, in the State Mining Engineer's *Report* for 1896.

"The Black Reef proper has, in general, only at the outcrop, or not far distant therefrom, the character of a reef in the common sense of the word.

"In a quartzitic chloritic matrix are found a small number of quartz pebbles, which never reach more than a medium size or exceed that of an ordinary walnut. At greater depth the pebbles become more rare, and the whole reef adopts more the character of a coarse-grained quartzite.

"Besides the (mostly dark gray coloured) quartz, a great many pebbles and fragments of flint and black lydian stone are found, and also, though seldom, small pieces of halleflinta.

"Near the footwall the reef contains very numerous fragments of the underlying diabase, as shown by the microscope. Sometimes they are sharp-edged, and therefore it is impossible that they could have been carried by the water over a long distance. These diabase fragments are, as a rule, strongly decomposed, now and then their place being completely filled up by a white kaolin-like mass. Pyrites appear rather amply, but always in small veins. It is only very seldom that it is spread evenly all over the reef.

"Crystals of pyrites are found rather often, many times as pseudomorphs of limonite to pyrites."

The amygdaloidal diabase forms the footwall of this reef, and the hanging wall consists of a dark quartzite, or sometimes of a schist, which Mr. Kubale has shown by microscopical investigation to be a grauwacke, and to be allied to the quartzite.

In this district the Black Reef is found conforming to channels or troughs in the bed of diabase; these troughs have a general east and west direction, and they maintain a marked regularity in width, direction, and succession, as illus-

trated in the Eastleigh mines. At the crests of these troughs the reef is generally poorly represented, but in the hollows it is developed to its greatest extent both in thickness and in auriferous character. The hanging wall does not conform to the channels in the footwall diabase to the same extent as does the footwall. The average dip of the reef in this district is about 10°.

Krugersdorp District.—In this district the Black Reef has been opened up principally in the Midas Estate mine, situated on the farm Luipaardsvlei, about 10 or 11 miles south of Krugersdorp. In this mine the immediate hanging wall is a bed of "shale," which is stated often to carry gold; above this there is an alternating series of quartzite and schists such as is given on p. 32. In all respects of occurrence and of character the reef is identical with that in the Klerksdorp district. The irregularity in assay value is marked, two consecutive samples giving in most cases widely different results. The results of the reduction processes have shown that a greater proportion of the gold recovered is obtained in the cyanide process, whereas with the banket of the Witwatersrand Beds it is usual to recover in the mill twice as much value as by the cyanide. The average dip in this district is about 10° or 12°.

In consequence of the absence of the amygdaloidal diabase the reef is here bedded on quartzite.

Black Reef District.—In this district, which includes the Orion group of mines, the Black Reef is a hard siliceous banket, of which the quartz pebbles are usually small. It varies considerably in thickness from an inch up to as much as 14 feet; the lowest portion of the banket is generally well mineralised with rolled and crystalline pyrites, so that pieces from it often have a specific gravity of over 3.00, that of normal banket being about 2.63, and it contains, in addition, fragments of the amygdaloidal diabase on which it has been deposited. Between the banket and the

diabase there is a soft, dark, ferruginous clay near the surface, which in places is highly auriferous, and which has given the name to the "Black" Reef; in the lower levels this clay becomes a hard and well-mineralised slate. The immediate hanging wall of the reef is a hard quartzite. In the cement, nickel and cobalt have been found, and in one place a limited occurrence of pure graphite was discovered. In this district also zones or shoots of considerable thickness and better grade follow the axes of longitudinal troughs, which generally run in an east and west direction. The gold is very unevenly distributed throughout the banket, and in this respect the better shoots differ from the poorer only in having a larger range of assay. In the reduction processes more gold is obtained from the cyanide than from the mill, this being probably in consequence of the pyritical nature of the ore, the amount of pyrites being about 10 per cent by weight of the rock milled.

The average dip in this district is about 8°, though along the extension in a westerly direction through the Vesta mine it is about 18°.

CHAPTER IV

THE MAIN REEF SERIES

THE Main Reef Series includes that group of banket beds which lies near the base of the Witwatersrand Beds, and from which the great bulk of the gold output of the Transvaal Colony is being obtained. The map facing p. 100 shows the most important portion of the extent of this series. It is characteristically developed in the district to the south of Johannesburg, so that an average section of the series at this centre may be regarded as a standard section. The following principal beds occur in order, commencing from the lowest :—

The Main Reef.
The Main Reef Leader.
The South Reef.

The Main Reef generally consists of several beds of banket with an aggregate thickness of about 7 feet, separated from one another by layers of quartzite with an aggregate thickness of about 3 feet, making in all a total thickness of about 10 feet. There is less of the quartzite in the top portion, while, towards the bottom, narrow stringers occur, which make the footwall indefinite, and to include which a total thickness of as much as 17 feet has sometimes to be measured.

The pebbles are well rounded and of true pebble shape, not spherical, the average size being close on an inch in length, and the extreme sizes from that of a pea up to two inches. They are chiefly of white quartz with a smaller

proportion of black quartz. Where the reef is compact they are placed thickly and uniformly in the cement, but not packed tightly, and they lie parallel to the bedding.

The cement or matrix which fills up the interstices between the pebbles is chiefly of siliceous and chloritic material, containing in addition a small percentage of iron pyrites.

The average gold content of this reef is low and the range of assay small. In the Crown Reef mine, out of a total width of 10 feet of banket and quartzite, the upper 4½ feet assay about 5½ dwts., and the less compact lower portion, 3½ feet thick, assays 3 dwts. to the ton: in this case the quartzite partings assay almost as much as the banket. The following values of the upper portion of this reef were obtained in the Wemmer, where the total thickness is about 8 feet:—

Half-year ending	Levels.	Feet driven.	Thickness. Inches.	Assay. Dwts. per ton.
31st Aug. 1895 . .	2nd and 7th	66	66½	7.8
29th Feb. 1896 . .	7th	53	21.92	9.11
31st Aug. 1896 . .	7th	9½	48	3.92
	Average	.	. 40	7¾

This reef is separated from the overlying Main Reef Leader by about 3 feet of quartzite, and, in most cases, by a strong parting in the footwall of that reef, accompanied by some soft schists. It is the experience in the Robinson mine that the Leader is richer where this parting is well developed, and in the Ferreira that, where it is not well developed, the Leader is poorer and the upper portion of the Main Reef richer, its value over some lengths being as much as 12 dwts. per ton for a thickness of 20 inches.

In some places in the Ferreira and Robinson Deep these two reefs are so close together, and the Main Reef so much better in grade, that the identity of the two is difficult to determine.

In the upper levels of the Wemmer the thickness of quartzite between these two reefs has increased from an average of 3 feet in the western portion of the Ferreira to about 60 feet.

This great thickness, however, gradually lessens on the other side, till in the City and Suburban it is again back to 3 feet. In the lower levels of the Wemmer itself this has also occurred, as shown in the following statement :—

	Main Reef to Main Reef Leader.	South Reef to Main Reef Leader.	South Reef to Main Reef.	Dip.
	Feet.	Feet.	Feet.	
At surface . .	63	44	107	81°
1st level . .	56	53	109	81°
2nd „ . .	56	50	106	73°
3rd „ . .	63	46	109	66°
4th „ . .	54	54	108	56°
5th „ . .	54	53	107	58°
6th „ Salisbury dyke crosses				50°
7th „ . .	36	82	118	40°
8th „ . .	14	122	136	35°

The pebbles of the Main Reef Leader have a larger average size than those in the Main Reef, and they have been found up to 5 inches in length. White and dark-coloured quartz pebbles are in about equal proportions, and occasionally a red tinge is seen; some of the smaller ones have a clear pale blue colour. They are usually frozen hard into the quartzite of the hanging wall, but are separated from that of the footwall by the strongly developed bedding plane, which along the Central Rand is characteristic of this reef. This band of soft schists is so well marked, that a contractor driving on the Main Reef Leader generally gets less per foot driven than if he were driving on the South Reef.

The Main Reef Leader has a very fair value: in the Ferreira the average thickness and assay value are about 16 inches and 30 dwts. per ton respectively; in the Wemmer the following values have been obtained :—

Half-year ending	Levels.	Feet driven.	Thickness. Inches.	Assay. Dwts. per ton.
29th Feb. 1896 . .	3rd to 9th	685½	19.86	28.26
31st Aug. 1896 . .	3rd to 9th	1582	17.04	34.47
	Average .		17.75	32.5

The general nature of the banket of this reef is identical with that of the Main Reef, though generally the percentage of pyrites is somewhat higher, without, however, materially affecting the specific gravity of the banket, which Mr. Franklin White has determined to be about 2.7 for the Main Reef Series. This pyrites, while chiefly occurring as small crystals disseminated throughout the cement, is sometimes seen as small rolled pieces, more especially near the footwall.

In the eastern portion of the Robinson mine the pebbles in this reef are peculiarly arranged; they lie with the bedding, and are packed closely, being so wedged in amongst themselves as to allow of little cement between them. Where this occurs the reef is found to be poorer. By microscopical examination this peculiar arrangement is seen to be due to compression.

The South Reef overlies the other two, and is separated from the Main Reef by a bed of quartzite which, including the Main Reef Leader, is about 72 feet thick. It is seen from the table above, that in the Wemmer, though the distance between the Main Reef and Main Reef Leader varies considerably, that between the South Reef and the Main Reef, taking into account the decreasing dip, remains much more constant.

The pebbles of this reef are smaller than those of the other two, though in all other respects the banket is very similar to that of the Main Reef Leader. A piece of very pyritic ore from this reef in the Village Main Reef had a specific gravity of 3.170. In the Jubilee mine there is, west of the shaft on the fifth level, a curious pyritical and sparry vein overlying this reef. This reef is not often compact; it is

more usual to find it consisting of two or three leaders separated from one another by bands of quartzite, so that while the total thickness of banket does not often exceed 4 feet, the size of stope necessary to include it all reaches as high as 8 feet in the Ferreira and 13 feet in the Bonanza.

Where there are three leaders, the upper one is usually of low grade, the middle one rich, and the lower one often rich but irregular; where there are only two, the upper one is of low grade and the lower one rich. It is usual to find a parting—*i.e.* a well-developed bedding plane—under the richer one, with which a small thickness of fissile quartzite is often associated.

In this portion of the Rand, this reef is the richest one: in the Ferreira it will average about 29 inches of clean banket with 70 dwts. in assay value; in the Wemmer the following averages have been obtained:—

Half-year ending	Levels.	Feet driven.	Thickness. Inches.	Assay. Dwts. per ton.
29th Feb. 1896 . .	8th to 10th	967	23.48	93.52
31st Aug. 1896 . .	8th to 10th	773	21.40	56.19
	Average . .		22.5	78

In addition to these principal reefs, there are other unimportant banket beds. To the north, separated from the Main Reef by quartzite 80 feet thick, there is one called the North Reef, which is about 12 inches thick, and poor. Between the Main Reef Leader and South Reefs, generally nearer the latter, there are several narrow beds of small-pebbled banket, of which the largest one is called the Middle Reef; this reef is about 12 inches in thickness and 4 dwts. per ton in assay value. Above the South Reef there is a large extent of small bastard reefs; in No. 2 shaft of the Robinson Deep these were struck at a depth of 1636 feet, from which point they continued down to the South Reef, which was struck at 1806 feet, as shown in the appended section (Fig. 13). In No. 1 shaft of the same mine these

bastard reefs were struck at 1891 feet, and the South Reef at 2385 feet.

These reefs consist of small pebbles, chiefly of white quartz, thinly scattered in the quartzite.

The whole sequence of the Main Reef Series, with the exception of the North Reef, as it occurs south of Johannes-

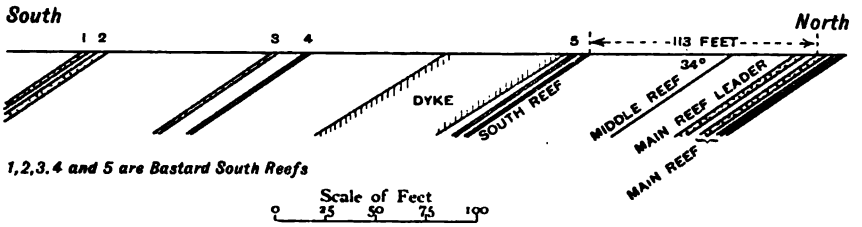


FIG. 13.—Section of the Main Reef Series in No. 2 shaft, Robinson Deep. Depth of South Reef from surface, 1806 feet.

burg, is shown in the following statement and section obtained from No. 2 shaft of the Robinson Deep :—

Depth below surface. Feet.	Reef.	Thickness. Inches.	Assay. Per ton.	Remarks.
1636	Bastard	24	...	Small pebbles, scattered, some pyrites.
1640	,,	36	...	Small pebbles, scattered, some pyrites.
1700	,,	24	...	Small pebbles, scattered, some pyrites.
1712	,,	12	...	Small pebbles, scattered, some pyrites.
1760-1800	Dyke			
1802	Bastard	24	0-10 grs.	Medium-sized pebble.
1806	South Reef	12	60 dwts.	
1855	Middle Reef	4	Trace	
1877	Main Reef Leader	36	12¼ dwts.	
1880	Main Reef	30	4½ ,,	
	,,	10	8½ ,,	
	,,	12	3½ ,,	

The general features of the Main Reef Series south of Johannesburg having been described, mention may be made of the following special occurrences. Zinc blende and

pyrrhotine have been found to occur, and more rarely galena and copper pyrites. In the Crown Reef mine, between the two central faults,—one on either side of the main incline shaft,—a number of auriferous veins of secondary quartz occur in and about the reefs, especially the South Reef. In these quartz veins visible gold has in some places occurred very thickly. In the Jubilee mine there is a case of what would appear to be sympathy between the gold content of the Main Reef and that of the Main Reef Leader. Just east of the shaft on the fifth level the former reef had improved in grade, whereas the latter, in the same place, had so deteriorated that the drive was turned to follow the Main Reef till, going east, this reef became poor, when the drive was turned back on to the Main Reef Leader, which had then regained its normal value.

From the Main Reef Series, in the Robinson Deep, No. 2 shaft, a large pebble was taken which showed a secondary growth of crystalline quartz on its surface.

Going east from the standard section at Johannesburg, the South Reef gradually approaches the Main Reef, and, with the exception of a short length centred around the Henry Nourse mine, the Main Reef Leader is close against the Main Reef. It is probably a coincidence that at the two places—the Wemmer and the Henry Nourse mines—where the Main Reef leader has receded farthest from the Main Reef, the dip of the series at the outcrop is greatest.

Another point noticeable going east as far as the Glencairn is the gradual improvement in grade of the Main Reef—an improvement which is only broken at the Henry Nourse. In the mines in the neighbourhood of the Wolhuter this is consequent upon the development of a footwall leader in the position shown in the following section (Fig. 14).

This leader is most conspicuous in the Wolhuter itself, where its average value is about 21½ dwts. per ton, and 7½ inches in thickness. In this mine it is characterised by carrying very large pebbles. Wherever possible, it is stoped with-

out taking the bulk of the Main Reef, from which it is separated by about 9 inches of quartzite. This leader is very irregular, and cannot be counted on to exist in the mines farther east, where, however, the bulk of the reef becomes of better grade.

In the Henry Nourse mine the characteristics of the three reefs, as exemplified at Johannesburg, are well maintained, as shown by the following brief description of the reefs in that

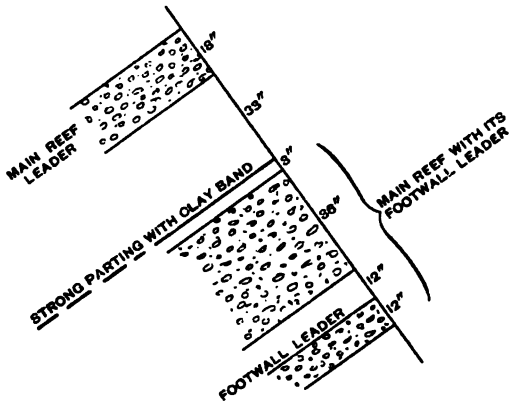
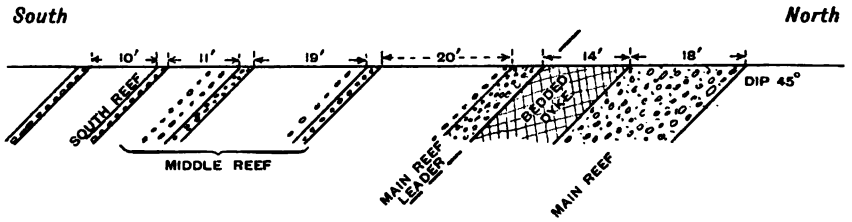


FIG. 14.—Section showing Main Reef Leader with the strong parting in its footwall and the Main Reef, in a stope on the 4th-level George Goch No. 3 mine.

mine. The South Reef is sometimes compact and sometimes composed of two or three stringers. When there are three, the top one rarely carries much gold, the middle one is best and most uniform in grade, and the footwall one is sometimes very rich and sometimes very poor. This reef is the most valuable in the mine. Its distance south of the footwall of the Main Reef Leader varies from 25 to 30 feet in the upper levels, which is equal to a true thickness of about 25 feet, whereas south of Johannesburg this thickness is about 65 feet. The Main Reef Leader is in this mine called the Middle Reef. It varies considerably in thickness and in value, and throughout the mine it will average about 20 inches thick, and 24 dwts. to the ton. The Main Reef is poor, and about 10 feet thick. It lies in the upper levels about 40 feet north of the

Main Reef Leader. It is at this point that the Middle Reefs assume larger dimensions, and begin to carry larger pebbles, as indicated in the section (Fig. 15), so that the South Reef



Scale, 1 inch = 30 feet

FIG. 15.—Section of Reef Series obtained from a bore-hole on the Nourse Deep Property.

becomes one of a series of banket beds, and is distinguishable only by its larger gold content.

Farther to the east, in the New Heriot mine, this distinction, though still noticeable, is less well marked, the South Reef averaging 21 inches thick and 30 dwts. per ton. The following section (Fig. 16) of the series obtained in that mine

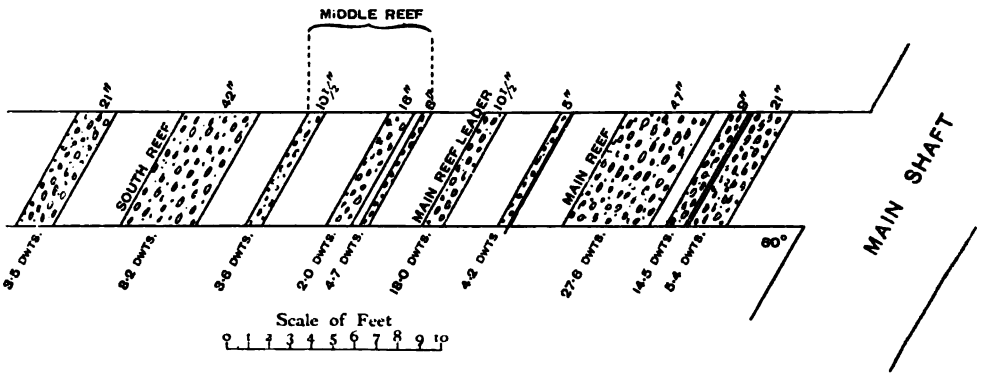


FIG. 16.—Section of Main Reef Series in a cross-cut south from the New Heriot shaft.

shows that the reefs have drawn much closer together, and that they are entirely included in a true thickness of about 33 feet.

Still farther to the east, in the Treasury mine, the South Reef can no longer be discriminated by its gold content from the Middle Reef, and the two reefs have closed up to form one large reef, as shown in the following section (Fig. 17).

This reef is generally low grade, but better grade leaders are sometimes found in it.

At this point the South Reef has assumed, with the Middle Reefs, the characteristics of the Main Reef of the Central Rand; and farther to the east, as long as it is distinguishable, it is known as the Main Reef.

The average character of the reefs from the Treasury to the Glencairn is well illustrated by their occurrence in the Rose Deep, of which the following is a brief description :—

The South Reef proper is here called the Main Reef. It overlies the Main Reef Leader, and is separated from it by a

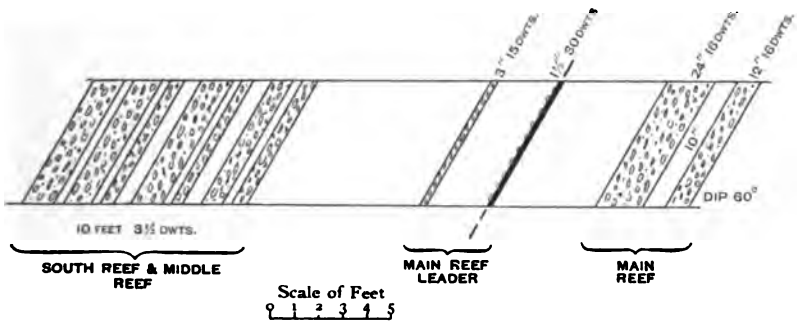


FIG. 17.—Section of the Main Reef Series in the Percy section of the Treasury.

thickness of quartzite which varies from 5 to 10 feet. On an average it is about 4 feet thick, and of lower assay value than the other two. East of No. 1 shaft its value is brought up by the occurrence of a leader in the hanging wall.

The Main Reef Leader is here called the Middle Reef. In the west and central portions of the mine it is very good: in its footwall there is a leader which is a few inches thick, but of high grade. It is estimated that throughout this mine this reef is 24 inches thick.

As it will be stoped with the Main Reef, its individual value has not been determined, but with the Main Reef it is estimated to supply stoping ore of about 15 dwts. assay value; of the two reefs, however, this one will probably provide

the richer ore. The Main Reef is here called the North Reef; throughout the mine it is estimated to average about $3\frac{1}{2}$ feet thick. It lies underneath the Main Reef leader, being separated from it by a thickness of quartzite which varies from 1 foot to 3 feet. This reef varies considerably in thickness in consequence of the intrusion of a bedded dyke, which pinches it so that in places it is only an inch or so thick; where the dyke does not occur, this reef is at times 7 or 8 feet thick. A drill core, from the 5th level of No. 1 incline, showed this reef to be 5 feet thick and to assay about 10 dwts. to the ton. In a bore-hole which was put down from the end of the south cross-cut on the third level of No. 2 shaft, the three reefs were included in a thickness of 20 feet or so.

There are, in addition to these reefs, a North Reef lying to the north of the Main Reef, and separated from it by a thickness of about 40 feet, and a bastard South Reef, separated from the South Reef proper by a thickness of about 50 feet. These two reefs are of no economic importance, and they may be correlated with similar reefs mentioned in the standard section to the south of Johannesburg.

This bastard South Reef is sometimes considered to be the South Reef proper, and that which has here been called the South Reef to be the Main Reef proper. Apart from the consecutive sections which have been given from the Henry Nourse to the Treasury, and from which it has been reasoned that the upper large and low grade reef, from that mine to the Glencairn, is the South Reef proper, there is the correlation to be made by referring the reefs in the area in question to the position of the slate parting. This slate parting occurs throughout, immediately underlying the Main Reef Leader, along a strong bedding plane which has been accentuated by the intrusion of a bedded dyke subsequent to deposition; reference to it confirms the correlation which has been adopted here.

The following section (Fig. 18) gives the complete series in the Geldenhuis Estate, and illustrates, in addition, the nature of reverse faulting :—

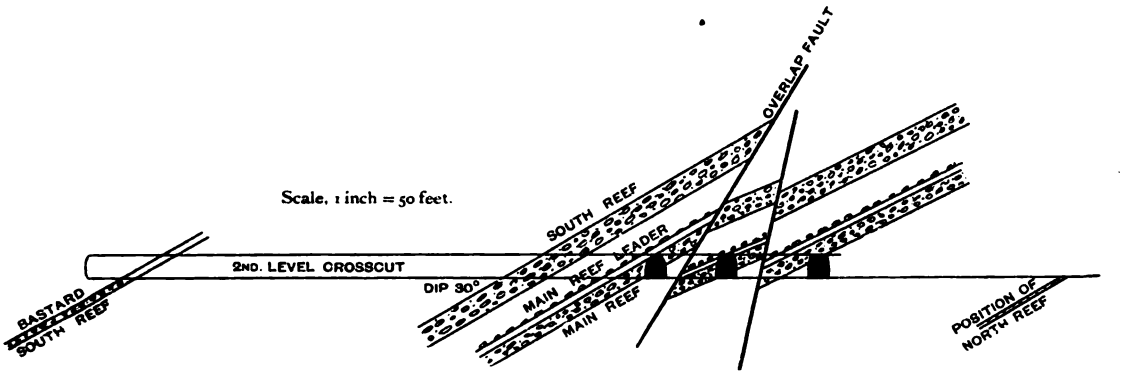


FIG. 18.—Section along the cross-cut south, on the 2nd level from the old vertical shaft of the Geldenhuis Estate.

The details of the Main Reef and Main Reef Leader, which are the only two reefs worked in this mine, are shown in the following sections (Fig. 19) :—

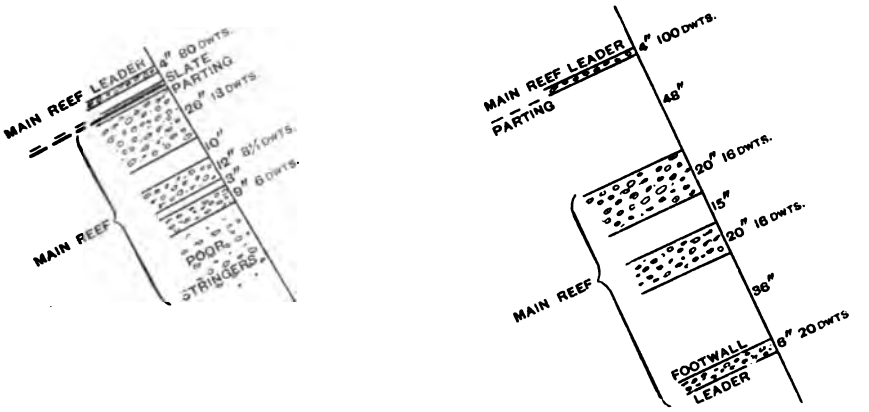


FIG. 19.

Section of reefs in the western portion of the Geldenhuis Estate.

Section of reefs in the eastern portion of the Geldenhuis Estate.

Farther to the east the series is well shown by the occurrence of the reefs in the Glen Deep No. 1 shaft, of which the following is a statement :—

Reef.	Depth.	Thickness.	Assay.
Bastard overlying Reef (South Reef) .	920 feet	6 inches	1 dwt.
South Reef (Main Reef) . . .	925 ,,	60 ,,	4 ,,
Main Reef Leader (Middle Reef) .	942 ,,	15 ,,	31 ,,
Main Reef (North Reef)*. . .	947 ,,	15 ,,	17 ,,
North Reef	978 ,,	7 ,,	10 ,,

The names in brackets are those by which the reefs are known on the property.

From this last statement it is seen that the Main Reef has here become considerably smaller.

On the property of the Witwatersrand Gold Mining Company, farther to the east, the Main Reef Series has become duplicated on the surface, by a dyke which is mentioned in Chapter V., the two outcrops being called the North and South Series respectively. Each in itself is a continuation of the full series; the section of the south series (Fig. 20) shows the relative positions of the different reefs:—

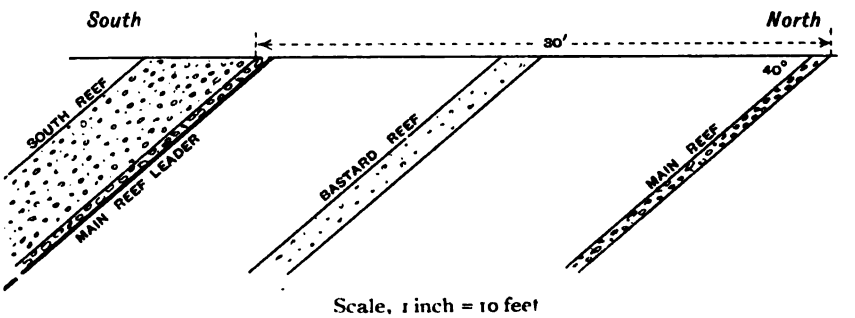


FIG. 20.—Section across the South Series, 3rd level, central section Witwatersrand mine.

The Main Reef is of no economic value; the Main Reef Leader and South Reef average, together, about 36 inches in thickness and 15 dwts. per ton in assay value. Of the two the Leader contains most of the gold; underlying it, there is the characteristic parting, and the Bastard Reef is probably a bedded dyke which, as in the May Consolidated, has enclosed some pebbles within its mass, giving it the appearance of a bastard banket.

From this point to the New Blue Sky, the Main Reef Series has been duplicated on the surface, though by a different dyke from that which effected the duplication mentioned above. The following description of the reefs in the New Comet mine represents very well the Main Reef Series in this area :—

In this property, as in the neighbouring ones, there are two reef series, which at the 1st level are 550 feet apart horizontally, and on the 5th level not quite 500 feet. These are called the North and South Reefs; they are duplicated outcrops of the Main Reef Series, the three principal reefs of which can be recognised in each.

The North Series varies from 3 feet to 14½ feet over all; the best part of it is a leader ranging from 6 inches to 16 inches thick, which in the centre of the mine is in the hanging wall, and towards the east, especially on the 1st level, in the footwall.

Forming the footwall of this leader there is a well-developed bedding plane or parting, underneath which there is a thickness of quartzite with a few pebbles in it, and under this again there is another piece of banket.

The leader is dark-coloured and good-looking, whilst the footwall piece of banket is poor and hungry-looking.

Underneath the whole series there is a strongly developed footwall.

A section of this reef series in the cross-cut 4th level is shown in Fig. 21, p. 82.

This includes all the intercalated quartzite, and shows over all a thickness of 14½ feet.

There was a strong parting underneath the upper 42 inches, and this banket contained medium-sized pebbles; below this there were 9 feet of dark quartzite with a few pebbles, then the footwall piece of 2 feet, which looked fairly well, and underneath this there was a strong footwall parting.

The upper band is to be recognised by its footwall, gold content, and its petrological character as the Main Reef

Leader, and the lower bands as belonging to the Main Reef; the South Reef is probably represented by the upper banket,

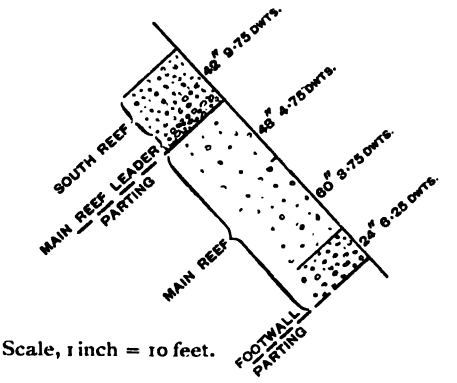


FIG. 21.—Section of North Series of reefs, New Comet mine, 4th level cross-cut.

which lies on the Main Reef Leader, in which case the section is very similar to that obtained on the Witwatersrand property.

As this reef series depends chiefly for its value on a small leader whose gold content varies considerably, it is very erratic in its value, as shown by the following figures:—

3rd level East for a length of 300 ft.: reef averaged 29 inches in width, and 3.81 dwts. per ton.

6th level West for a length of 386 ft.: reef averaged 31 inches in width, and 22.8 dwts. per ton.

On the 6th level this reef has a better value, the following being the averages for three consecutive weeks:—

Thickness	30	inches,	assay	35	dwts. per ton.
"	36	"	"	11½	" "
"	36	"	"	131	" "

These figures also indicate the varying assay of the reef; they were obtained from samples taken about 100 feet west of the shaft.

The South Series was struck in a bore-hole put down about 220 feet south of the position of the Phœnix shaft, at a

depth of 392 feet, where the bottom layer of 12 inches assayed 33 dwts. 6 grains.

This reef series consists of a compact body of good-looking banket about 24 inches thick, in the hanging wall of which, and separated from it by 24 inches of quartzite, there is another band, 12 inches or so in thickness, made up of small pebbles, and in its footwall there is another poor-looking reef which is separated from the main body by a bedding plane such as is usually found between the Main Reef Leader and Main Reef; below this again there will probably be more banket. This

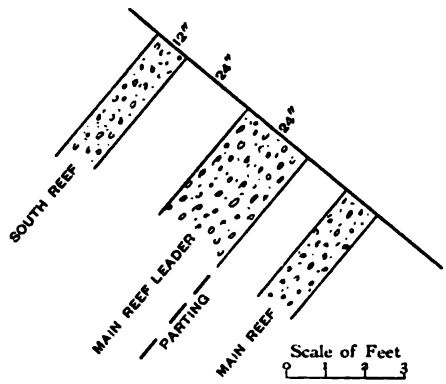


FIG. 22.—Section of portion of the South Reef Series exposed 4th-level Phœnix shaft, New Comet.

parting of soft schists underneath the main body was very well marked in the shaft where, in sinking, the miners took advantage of it; in the drive on the 4th level West it was also very noticeable, being, where approaching a transverse dyke, accentuated by the occurrence along its plane of a band of pyrites with some quartz; its occurrence throughout stamps the body of reef of which it is the footwall as being the Main Reef Leader, in which case the South Reef and Main Reef are represented by the upper and lower pieces respectively.

In the East level, about 150 feet from the shaft, the reef matter practically covered the face.

A section of this reef series at the 4th level near the shaft is given in Fig. 22.

In some places the thickness of quartzite separating these reefs is much greater, so that between the upper and middle piece it is sometimes as much as 6 feet, and between the middle and lower as much as 10 feet; such a separation is not, however, maintained over any great length.

At 513 feet in the shaft, the Main Reef Leader was 10 inches thick and assayed 94 dwts.; on the first level the ore was poor; on the 2nd level, for 250 feet driven, it averaged 30 inches in thickness and 16½ dwts. per ton in assay value.

In the eastern portion of the New Blue Sky, the two reef series have been opened up as shown in Fig. 37, p. 114. The South Series on the 2nd level is 350 feet south of the North Series. The North Series consists of two reefs called the Main and North Reefs; the Main Reef is divided into two sections, the hanging wall and footwall sections, which are thus described in the General Manager's report, 30th September 1896:—

"The hanging wall sections, like the North Reef, are very sparingly pebbled and carry only small quantities of gold; the footwall section produces a well-defined banket carrying a large white pebble. The assay value of this section is variable over a width of 6 inches, being as low as 4 dwts., and as high as 80½ dwts. per ton."

This footwall section has characteristics which make it probable that it is rightly correlated with the Main Reef Leader, in which case the sparingly-pebbled hanging wall section is the poor continuation of the South Reef proper.

The following description of the North Reef is also taken from the report mentioned above:—

"At the 2nd level, which has an incline depth of 241 feet from the surface, another cross-cut is put out to it, in which the total width of the reef was found to be 10 feet. The hanging wall sections, which are very sparingly pebbled, carry only a dwt. or two of gold per ton. The footwall sections produce well-defined banket, and vary in width from 6 inches

to 24 inches, ranging in value from 4 dwts. to 11½ dwts. per ton."

The position of this reef relative to the reef which has just been correlated as the Main Reef Leader, suggests that this is the Main Reef proper.

The South Series consists of one reef, which averages 48 inches in thickness and has a low assay value; of this thickness the footwall portion is the best. This reef may either be the Main Reef proper, in which case some more banket may be in the hanging wall, or it may be the Main Reef Leader, in which case the Main Reef proper has either not been developed, or it has been cut away by the dyke between the two series.

In the Van Ryn-Chimes district the Main Reef Series is represented principally by two banket beds, of which the lower one, called the Main Reef, is the larger and more persistent, the upper one being known as the Main Reef Leader; the section afforded by the occurrence of these two reefs in the Benoni mine is identical with that of the North Series of reefs in the eastern portion of the Blue Sky, and the petrological features in each case being similar, it seems safe to assume that they are exactly the same reefs; this opinion is confirmed by the stratigraphical position of these reefs, those of the Blue Sky lying about 125 feet south of the underlying shales of the Quartzite-Shale formation, and those of the Benoni, as judged by the occurrence in the Chimes Mines bore-hole, lying at a similar or slightly less distance. As stated in describing the Reefs of the Blue Sky, those of the North Series of that mine have characteristics which are those of the Main Reef Leader and Main Reef of the Central Rand respectively, and it is this correlation which may be extended to include the two reefs of the district under consideration.

Of these two the Main Reef is the more persistent; going eastwards it approaches the underlying slates, till in the New

Kleinfontein it rests entirely on them, and as far as development has proceeded farther east this position is maintained.

Though this reef can be traced to extend along the whole of the ground yet opened up, it has changed gradually from a large and low grade reef on Benoni, to a small and rich reef on Modderfontein, where in fact it has all the characteristics of the Main Reef Leader proper.

The Main Reef Leader, which on the Benoni is a compact body, has broken up on the Van Ryn Estate into several stringers of uncertain and irregular value; and still farther to the east this reef loses all its characteristics and changes to a series of small-pebbled and poor stringers.

The South Reef is barely represented at all in this district; its deterioration going eastward from the Central Rand can be gradually followed till, in the East Rand mines, it is practically a bastard Reef. In the Chimes mines bore-hole one of the series of Bastard South Reefs was encountered 4½ inches thick, assaying only traces. This reef correlates with that which has been shown in the Geldenhuis Estate section, Fig. 18.

A section of the reefs in the Van Ryn West has been given on page 52. The following description and sections of the Main Reef Series in the Van Ryn Estate show the breaking up of the Main Reef Leader, Figs. 23 to 25.

The Main Reef Series in the Van Ryn Estate forms a body of banket which lies immediately on the slate, and is known here as the Main Reef.

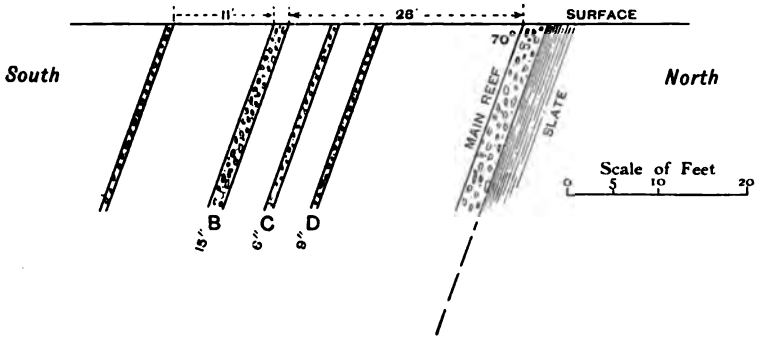
It varies in width and value as under :—

On the	2nd	level,	from	1	inch to	60	inches, and from	Nil to 373 dwts. per ton.
„	3rd	„	„	1	„	90	„ „	Nil to 203 dwts.
„	4th	„	„	1	„	48	„ „	Trace to 144 „

The averages on the 4th level, from 1500 feet west of the Main shaft to 500 feet east, are 16¼ inches and 25½ dwts.

respectively. Overlying the Main Reef there is a series of leaders from which high assays are sometimes obtained.

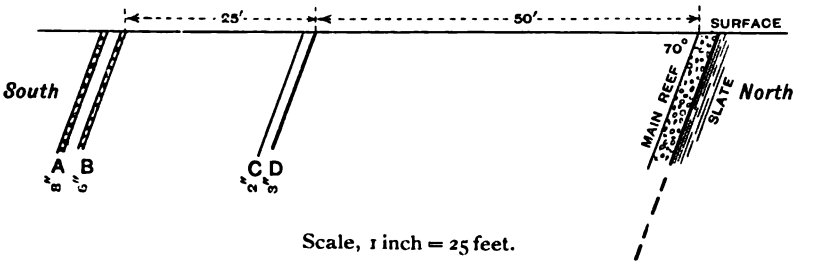
On the 1st level east of No. 4 shaft these leaders have been opened up by a separate drive, which lies about 15 feet



B	Upper portion	12 inches.	15 dwts.	12 grs.
	Lower ,,	3 ,,	39 ,,	17 ,,
C		6 ,,	13 ,,	23 ,,
D		9 ,,	6 ,,	3 ,,

FIG. 23.—Section 550 feet west of No. 4 shaft, Van Ryn Estate.

south of the Main Reef drive. It is along this stretch that they maintain a good value, which, by cross-cuts on the lower levels, is shown to extend in depth.



A	8 inches.	43 dwts.	12 grs.
B	6 ,,	90 ,,	8 ,,
C	2 ,,	1 ,,	0 ,,
D	3 ,,	Trace.	

FIG. 24.—Section at No. 4 shaft, Van Ryn Estate.

It would appear from these sections that the leader which carries the gold is not always at the same distance from the Main Reef. In the lower levels it lies at an average distance of 10 feet to the south.

In the chapter on the different banket series it has been stated that the Nigel Reef in the Nigel district presents characteristics which make its correlation with the Main Reef in the Van Ryn district extremely probable. In the Heidelberg Syncline that reef on the farm Rietbult which has been

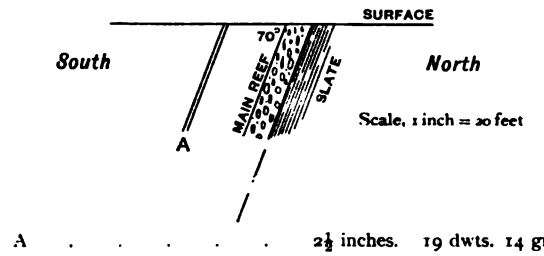


FIG. 25.—Section at Main Shaft, to the east of No. 4 shaft, Van Ryn Estate.

correlated with the Nigel Reef must also be correlated as the Main Reef, and it seems more than likely that the reef worked in the Heidelberg-Roodepoort mine is an extension of the same reef.

West of Johannesburg to the Witpoortje Break, the features of the Main Reef Series which are found south of Johannesburg are comparatively well maintained. The South Reef is separated from the Main Reef by an average thickness of about 70 feet, and the Main Reef Leader by about 3 feet.

The South Reef is the richest reef throughout, though it becomes much smaller.

The Main Reef Leader gradually deteriorates, whilst the Main Reef maintains its low grade until in the Roodepoort district it is in places and to a limited extent good enough to work. There is, however, an extent of ground between Langlaagte and Florida where the reefs are uniformly, at the outcrop, of a lower value than in the Johannesburg and Roodepoort districts on either side. There seems no doubt, however, but that with economic reforms the Main Reef Series in this area will prove payable. The relative positions of the reefs in the Langlaagte United are shown in the section

on p. 117. The following are some particulars of the reefs struck in the fifth level cross-cut of that mine :—

The North Reef consisted of three narrow beds of banket, each about 6 inches thick, interstratified over 6 feet of quartzite, and assaying from 4 to 6 dwts. per ton.

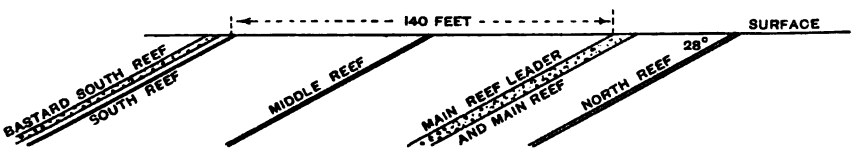
The Main Reef was a well-defined body of banket about 6½ feet thick, but of low assay value.

The Main Reef Leader was about 18 inches thick.

The Middle Reef was a small-pebbled banket bed about 18 inches thick, which only assayed traces. In one intersection this reef was broken up into several poor stringers.

The South Reef was about 36 inches thick and averaged about one ounce in assay value.

The relative position of the reefs in the Roodepoort district is illustrated in the following section (Fig. 26), which



Scale, 1 inch = 70 feet.

FIG. 26.—Section of the Main Reef Series obtained in the West Roodepoort Deep.

has been made from data furnished by bore-hole results and development work in the West Roodepoort Deep.

In this district the South Reef is a narrow banket bed about 3 inches thick, assaying several ounces. Its pebbles are usually small. It is worked in all the mines, and is the mainstay of the district.

Closely overlying it, it is usual to find a larger thickness of fine-pebbled conglomerate which has no value. In the Durban-Roodepoort mine it is considered that the South Reef is better when this bastard reef is not developed.

The Middle Reef is a fine-pebbled conglomerate of small width, and of no economic value.

The Main Reef and Main Reef Leader are in contact.

is well shown by the following particulars of its occurrence in the Lancaster. In the north cross-cut, 560 feet level, No. 1 shaft, at 75 feet from the South Reef, the lower portion of this reef was struck, assaying, over a thickness of 25 inches, 81 dwts. 6 grs. Of this width the upper 15 inches assayed only a few dwts., the bulk of the gold being in the lower 10 inches, as shown by the following results:—

	Cross-cut. East Side.	Cross-cut. West Side.
15″ hanging wall .	3 dwts. 3 grs. per ton.	12 grs. per ton.
10″ footwall . .	155 „ 6 „ „	278 dwts. 10 „ „

Lying closely above it, there were two poor bands of banket, of which the upper was 16 inches thick and the lower 10 inches. It is found in the Luipaardsvlei Estate that though the footwall portion of the reef is the best, the upper portion, when it carries large pebbles, which is more rarely the case, is equally good, the gold apparently following the larger pebbles. This occurrence of better ore in the upper portions has been noticed in other mines, but it is not well maintained; it would represent the position of the Main Reef Leader of the Central Rand.

The South Reef is a narrow and rich reef overlying the Botha's Reef, and separated from it by a thickness of about 20 feet. Its average thickness is about 2½ inches, and its value is generally to be measured in ounces. It is characterised by the flat shape of its pebbles. These are small oblate spheroids, and in many cases the encircling edge which runs between the top and bottom is, though rounded, comparatively sharp. Though this reef is rich, the mines working this series rely chiefly upon the Botha's Reef.

In addition to these two reefs some fine-pebbled banket beds occur between them, but these are of no value.

Bastard Reefs.—These have been repeatedly referred to, and may now be more fully described. Generally speaking, a bastard reef is one which, occurring with any series, is for that series undoubtedly poor-looking.

S

AL

EAST

E N

VAN RYN-CHIMES DISTRICT.

Scal

Scale across the formation
100 feet = 1 inch.

M A

MAIN REEF

MAIN REEF LEADER

R

H

Ferreira	87°
Wemmer	[illegible]
Cinderella	40°
New Blue Sky	40°
Benoni	36°
Chimes West	30°
New Kleinfontein	40°
Van Ryn West.	67°
Van Ryn	66°
Modderfontein	37°
Modder. Exten.	10°
Klipfontein	15°

With the Main Reef Series the average banket of the three principal reefs consists of pebbles, on an average about an inch long, pretty uniformly distributed throughout a matrix or cement which shows a good deal of iron pyrites. The greater number of these pebbles are of white quartz, but a considerable number of dark or black quartz pebbles are always seen, and occasionally some vari-coloured tints, light blue and more rarely pink, occur.

With this series a bastard reef means the occurrence of a bed of very small pebbles, or of ordinary-sized pebbles, sparsely distributed through a bed of quartzite.

The former occurrence is the more frequent; it is well shown by those small-pebbled banket beds which overlie the South Reef, and which, as they occur in the Robinson Deep, are mentioned and shown on p. 73. In this series also, the Middle Reef proper, where it is distinct from the South Reef, consists of small pebbles. The second occurrence of bastard banket in this series, viz. that of ordinary-sized pebbles, sparsely distributed through a bed of quartzite, is principally found in connection with the poorer portions of the Main Reef. These portions of that reef contain pebbles of an ordinary size, which are almost entirely of white quartz, whilst but little pyrites is to be seen.

Experience has proved that such bastard reefs or such beds of bastard banket in the Main Reef Series do not contain any material amount of gold.

In order to show comprehensively the relation of the three principal reefs of the Main Reef Series to each other, the subjoined diagram (Fig. 29) has been prepared, in which the Main Reef has been taken as a base line, and the distances from it of the other reefs has been plotted. It must be remembered that this plan is purely diagrammatic and in no wise indicates the strike of any of the reefs.

CHAPTER V

DISLOCATION: DYKES AND FAULTS

DISLOCATION may occur, from the action of the dynamic forces induced by the shrinkage or contraction of the earth's mass, both at the moment of the formation of a fracture and subsequently. It is brought about either by movement along the plane of fracture, which constitutes a "fault"; or by separation of the fractured ends by intrusive matter, which constitutes a "dyke"; or by a combination of both.

The dislocations along the Witwatersrand, in relation to the direction of stratification, may be divided into the following three groups:—

1. Longitudinal dislocations, which run more with the strike of the formation than across it (strike faults).

2. Transverse dislocations, which run rather across the strike of the formation than with it (dip faults); and

3. Bedded dislocations, which separate the planes of the reefs.

Whilst any dislocation rarely belongs entirely to one of these groups, but in some part of its extent may be classed as one or another, yet those which are met with along the Witwatersrand show evidences that they were developed in response to dynamic movements along well-maintained directions, and in systems which allow of the above classification.

In faulting, one or both of the fractured extremities of the beds may have been shifted; for convenience the one which, in following the reef, is first met with is considered to have

been relatively undisturbed, though such may not necessarily be the case.

Dislocations are further and more generally described by the nature of their throw. Where, going from one terminal to the other, the direction towards the greater angle between reef and fault planes is followed, the throw is termed

FIG. 30.—Normal and reverse faults.

"normal," and where the smaller angle is followed, the throw is called "reverse." These two conditions are represented diagrammatically in Fig. 30, whilst the general type of overlap faults when the beds are steeply inclined is shown in Fig. 31.

The demonstration of the existence of dislocations in series can also be shown by dividing them into right-handed

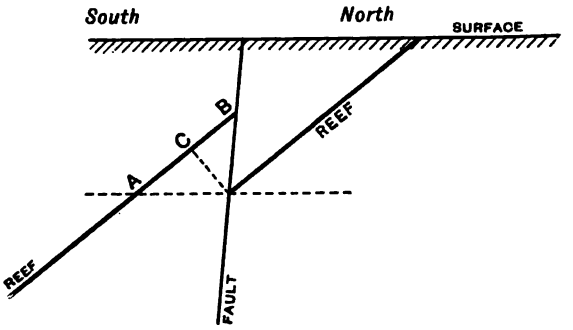


FIG. 31.—Diagram of fault.

and left-handed dislocations, according as the direction of the faulted extremity lies on the right or left hand. This is exemplified in the statement on p. 121, in which the dislocations are uniformly right-handed, indicating a series, but not uniformly normal. A right-handed throw may be normal or

reverse, and the same is also the case with a left-handed throw.

With a reverse throw, the plane of the faulted portion of reef is caused to overlap that of the undisturbed portion, so that in a certain area of ground the reef occurs twice. It is seen from points 5 and 6 on p. 115 that reverse faulting generally occurs with longitudinal dislocations, so that the occurrence is generally best illustrated in a vertical section across the formation, such as is shown in Fig. 32.

It is usual in most mines to regard all the faulted portion of the reef which is above the level of the undisturbed extremity, such as the length AB in Fig. 31, as being an overlap, though such would only be the actual case if the reef were vertical; the true amount of overlap is represented by BC. On p. 118 a detailed statement is given of all the faults and dykes which cross the outcrop of the Main Reef Series from the Crown Reef to the George Goch.

Transverse Dislocations.—These are most noticeable in the Van Ryn-Chimes district and in the Krugersdorp district. In the former district the strata have been broken by a number of faults which, being similar in their action and occurrence, have most probably resulted from one particular and local disturbance; this series is shown in the statement on p. 121. From the intersection of No. 1 dyke to that of No. 9 is a distance of 5500 feet in a direction E. 15° N.; in this distance the reef has been thrown an aggregate distance of 1298 feet to the south; in addition, there is a change of strike in the same direction as the reef is thrown, amounting to 6°; so that altogether, by right-handed throws to the south and by change of strike, the reef at its eastern termination against No. 9 fault is about 2325 feet south of the position it would have occupied had the strike in the western portion of the Van Ryn West continued undisturbed.

Referring to the statement, it is seen that the one reverse fault is the largest; about it all the others were probably

formed, and it, more than the others, is the direct result of the disturbance.

In the Krugersdorp district there is a similar occurrence of step faults in series, accompanied by a change of strike in the same direction as the reef is thrown. Going west from the property of the French Rand mine the reefs arrive, by change of strike and by faulting, farther and farther south of the line continued from that mine; had they maintained that direction they would have passed north of Krugersdorp, instead of which they have gradually slipped to the south, till at the West Rand mines they are 8000 feet out of line. Farther west, on the farm Waterval, this occurrence gives way to a radical change of strike from a direction east and west to one north and south through the Randfontein mines. This change of strike probably represents, more directly than the dislocations, the result of this local and particular disturbance.

The statement, on p. 122, of the dislocation of the different reefs on the Lancaster property, shows how they were affected in common with the rest of the formation.

It may be stated that this series of faults does not start in the French Rand mine; it is found eastwards beyond the Witpoortje Break in the Durban Roodepoort, Princess Estate and Banket mines.

In the Central Rand there is no regular series of transverse dislocations; from the statements given on p. 118 it can be shown that these dislocations around Johannesburg practically balance one another. This balancing is not only noticeable in general but in detail; the faults are often found in pairs of opposite tendency, one a right-handed and the other a left-handed throw; this is noticeable in the following cases:—

The two faults, one on either side of the Crown Reef Main Incline, Nos. 1 and 2 in statement on p. 118.

The two faults near the eastern boundary of the Robinson, Nos. 11 and 12.

The two faults east and west of the City and Suburban Main Incline shaft, Nos. 29 and 30.

The two faults near the western boundary of the Meyer and Charlton, Nos. 32 and 33.

And, on a larger scale, the George Goch and Metropolitan faults, Nos. 43 and 46.

In each of these cases the strikes of the two faults are more or less rapidly converging, so that at some point not far removed the two faults will tend to neutralise one another. That this is so is well instanced by the depths at which the reefs were struck in No. 2 and No. 3 shafts of the Nourse Deep: in No. 2, at a distance of 820 feet from the outcrop, the South Reef was struck at about 1433 feet, and in No. 3, 540 feet from the outcrop, at a depth of 1400 feet. The reef in No. 3 shaft had been thrown down about 390 feet by the Metropolitan fault, whilst before No. 2 shaft was reached this fault had approached and united with that from the George Goch, making with it a resultant downthrow in this shaft of only 130 feet.

In the mines around Roodepoort, as stated before, the series of step faults so well established around Krugersdorp commences. These faults strike in a south-westerly direction and throw the western terminations of the reef to the south. In No. 1 block of the Durban Roodepoort there are three such faults, which, crossing the outcrop, result, in a length of 1200 feet, in throwing the reef going west about 120 feet to the south, and in the main shaft of No. 2 block similar faults have been encountered. In the Princess Estate the Western portion of the mine is separated from the main portion by a large dyke 50 feet thick, which, striking in a south-west direction, throws the western extremity of the reef about 1200 feet to the south. The Banket mine, which adjoins the Princess on the west, is divided into two by a fault which throws the western portion about 600 feet south.

Farther to the west the Witpoortje Break is encountered;

against this the Main Reef Series is very much broken, pieces like those in the Gipsy Gordon and Bohemian properties being detached. The actual terminations on either side of this break have not yet been definitely fixed, but from the Banket mine to Grey's Mynpacht is a distance of about 3 miles in a north and south line.

It is seen from the statement on p. 118 that the Grahamstown dyke has been classed as longitudinal, though at its intersection with the outcrop of the Main Reef Series it strikes almost directly across the formation. This is because its course south of the outcrop has been proved to extend for a long distance in an average easterly direction. There is, between the Treasury and Geldenhuis Estate, a heavy dislocation which throws the outcrop in the latter mine about 850 feet south of its position in the Treasury. This dislocation has taken place along a fault which strikes between the two terminals in a N.N.E. and S.S.W. direction. There is at the same place a dyke which strikes north and south, and which has been met with in the Treasury mine. It is generally considered that this dyke and fault are one and the same occurrence, but this is not likely, because in the western portion of the Geldenhuis Estate the fault has been pierced through without finding any dyke; further, the strikes of the two are different. This Geldenhuis Estate fault is slightly reverse in the nature of its throw. In view of the fact that a larger proportion of the reverse faults are longitudinal, and that two of the principal longitudinal dislocations —the Grahamstown dyke and the Simmer dyke—have, where they cross the outcrop of the Main Reef Series, a transverse strike, it becomes quite likely that the Geldenhuis Estate fault is a similar occurrence; in which case, as it extends south, it will turn to the west and cause an advantageous upthrow in the Jupiter and other deep level properties.

Though along the Central Rand there is no extensive series of transverse faults, there are many dykes which cut

almost directly across the formation, but these rarely disturb it to any extent.

The Robinson mine is divided very equally into two parts by a large dyke which is about 230 feet thick and has no perceptible throw. This dyke dips about 70° to the west, and its outcrop can be traced on the surface to extend in a north and south line to a position on the south-western corner of the Robinson Deep. It is composed of a coarsely crystalline igneous rock similar to that of the longitudinal dykes which cross it. In the Meyer and Charlton there is a dyke about 94 feet thick, which strikes in a north-easterly direction and is practically vertical. It throws the eastern beds about 80 feet to the north. In the George Goch there is another dyke which does not throw the reefs.

In the Henry Nourse there is, in addition to others, a large transverse dyke about 240 feet thick, which has comparatively little disturbing effect on the strata.

Near the western boundary of the Witwatersrand property there is evidence to show that a considerable transverse dislocation exists, and another must occur between the western and the central portions of this property.

It is interesting to note that between the central and the eastern sections of this property there is another occurrence of a longitudinal dyke crossing the outcrop transversely.

According to the statement of the transverse dykes opened up in the immediate neighbourhood of Johannesburg, twenty-seven are accompanied by normal throws and one by a reverse throw. The average angle of dip is 83°, and the direction is distributed equally between east and west.

Longitudinal Dislocations.—There is in the Central Rand a very pronounced series of these dislocations, the most important of which are shown in the accompanying map. Apart from the dip of the formation, the position of the reefs in the deep levels depends upon the longitudinal components of dislocations. This being the case, those which strike east and

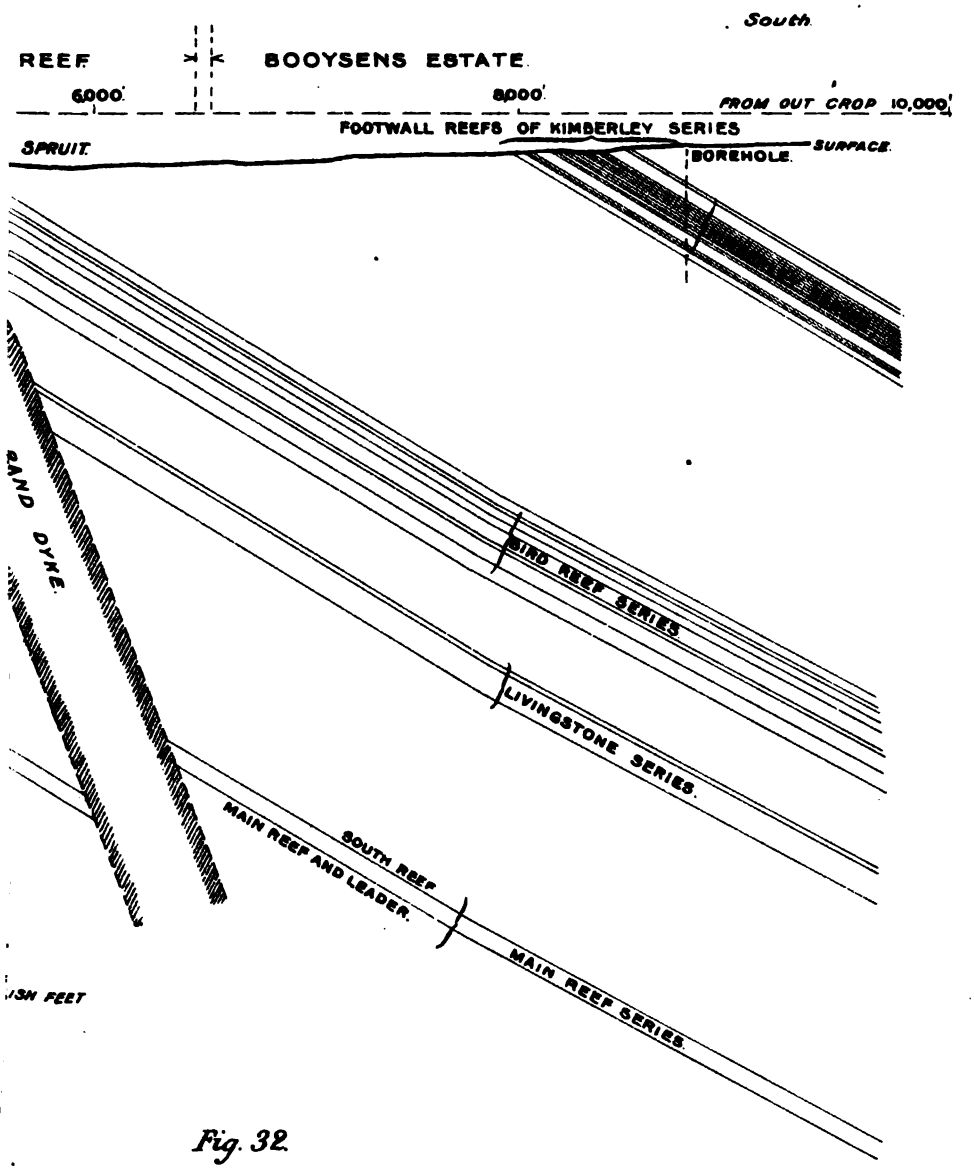

South
REEF
BOOYSENS ESTATE.
6,000'
8,000'
FROM OUT CROP 10,000'
FOOTWALL REEFS OF KIMBERLEY SERIES
SPRUIT.
BOREHOLE.
SURFACE.
SAND DYKE.
BIRD REEF SERIES.
LIVINGSTONE SERIES.
SOUTH REEF
MAIN REEF AND LEADER.
MAIN REEF SERIES.
ISH FEET

Fig. 32.

west along the formation have the greatest influence upon the deep levels, such influence decreasing when the dislocation becomes more transverse, until it vanishes entirely when the strike is north and south.

The following are some of the most important longitudinal dislocations :—

Ferreira-Crown Deep Dyke.—In the Ferreira this dyke has a strike about E. 10° N. and W. 10° S., and an average dip of 65° to the south. It crosses the outcrop of the Main Reef Series just at the boundary with the Wemmer. It is accompanied by an upthrow of about 50 feet on the south side.

This dyke also traverses the Crown Deep, where it strikes E. 18° N. and W. 18° S., dips 72° to the south, and has an upthrow of about 100 feet on the south side. It has an average thickness of about 120 feet in both properties. West of the Crown Deep it is in all probability represented by the dyke which has been found in the Langlaagte United, the effect of which upon the strata is shown on p. 117.

The resultant displacement by all the dykes and faults shown in that section is such that the position of the South Reef at the end of the cross-cut is about 450 feet vertically above the position it would have occupied had it continued uninterruptedly down from the surface. In consequence of this large upthrow there is a duplicated outcrop of the South Reef on the surface, as shown in the accompanying Dyke map.

The Grahamstown Dyke.—This dyke crosses the outcrop of the Main Reef Series between the Salisbury and Jubilee. It is there about 200 feet thick, dips about 86° to the east, and is responsible for a throw, south from the Salisbury, of about 230 feet.

From this point its course can be followed, first in a S.S.W. and then in a more westerly direction, across the Ferreira Deep in a line almost parallel to the north boundary, after which it crosses the Ferreira dam and then runs along the south boundary of the Crown Deep, after which, changing its

direction somewhat, it crosses the ground of the Langlaagte Deep to the south-east corner of the Paarl Central ground, beyond which it no doubt continues.

At its occurrence near the outcrop of the Main Reef Series it is accompanied by reverse faulting which, when it strikes east and west, becomes an upthrow on the south side. It passes north of the Robinson Deep ground, where the depths at which the reefs were struck in the shafts demonstrate, in view of the maintenance of the same angle of dip as occurs in the lower levels of the Ferreira, that between these points and the outcrop an upthrow must have occurred.

The course of this dyke follows very closely that of the Ferreira Crown Deep dyke. It has been traced for 3½ miles on the surface, and no doubt continues farther.

The South Rand Dyke.—From the south-west corner of the Robinson Deep a dyke about 500 feet thick has been traced running roughly parallel to the southern boundary of the farm Langlaagte for a distance of about 3 miles, beyond which it was observed to continue. It has a dip of about 70° to the south, and is accompanied by a considerable upthrow on the south side, which is shown by the duplicated outcrops of the Bird Reef, one occurring north and the other south of the outcrop of the dyke.

There are evidences to show that the three dykes just described belong to the same system: the igneous rock in each case is granular and coarsely crystalline, they are approximately parallel in direction, and they are all accompanied by reverse faulting.

George Goch Dyke.—There is a dyke about 130 feet thick running east and west through this mine. It outcrops about 100 feet south of the Main Reef Series, and dips about 75° south, intersecting the South Reef just below the fourth level. It is accompanied by a considerable upthrow on the south side.

Going east, this dyke gradually curves round to the north,

so that at no great distance it will have crossed into the foot-wall of the Main Reef Series. Its extension west has not been proved. This dyke is disturbed by all the other faults and dykes whose intersection with it have been opened up, so that it is one of the oldest disturbances yet met with in these mines. It is granular, and similar in appearance to the dykes before mentioned.

In addition, there is in this mine another fracture, running east and west, along which reverse faulting, resulting in the duplication of the Main Reef, has occurred.

This duplication extends at least 500 feet horizontally. It occurs down to below the third level, and is called the North Reef.

This fracture must dip about 55° or 60° to the south.

Metropolitan Fault.—This fault breaks the continuity of the reefs from the George Goch to the Henry Nourse. It strikes about E. 40° S., and dips 70° to the north-east, the reef from the George Goch being thrown about 480 feet south.

The Simmer Dyke.—This dyke crosses the outcrop of the Main Reef Series at a distance about 500 feet inside the eastern boundary of the Simmer and Jack property. At this point it has a strike E. 32° S., a slight dip to the north-west, and it throws the line of the reefs about 1200 feet to the north on the east side. A cross section would show this to be an upthrow on the south side.

To the south the strike soon alters to a more easterly direction, so that this dyke crosses the upper part of the No. 2 shaft of the Rose Deep and passes near No. 2 shaft Glen Deep, between which points it runs parallel with the formation. Still farther to the east its outcrop curves to the north and passes over the Glencairn ground to that of the Witwatersrand Company, where it enters between what are called the North and South Reefs of that property. These reefs are really duplications of the Main Reef Series caused by this fault. The upthrow on the south side must here

amount to at least 400 feet, and it increases going east. It finally crosses the North Series in the eastern portion of this property.

Knight's Longitudinal Dyke.—This dyke disturbs the South Series of reefs in the Witwatersrand mine without affecting the North Series. This fact shows that it was intruded before the duplication of the reefs into the two series was effected. This dyke crosses the outcrop about 750 feet east of the main incline shaft. It dips almost vertically, and undergoes the following changes in strike :—

From surface to 2nd level	N. 37° E.
„ 2nd level to 3rd level	N. 70° E.
„ 3rd level to position in main incline shaft .	S. 83° E.
In the shaft	E.

It is about 47 feet thick, and throws the eastern termination about 200 feet to the south, a right-handed reverse throw; when the dyke strikes east and west this throw becomes an equivalent upthrow on the south side.

East Rand Longitudinal Dyke.—This dyke runs between the North and South Reefs of the East Rand Proprietary mines. Its position in the Blue Sky is shown in the section on p. 114. West of this mine it gradually approaches the South Reef till, in the Angelo and Driefontein mines, it intersects that reef, the contact between the two running more or less horizontally at a depth of about 200 feet below the surface. Still farther to the west, on the property of the Witwatersrand Deep, by a change of strike this dyke crosses to the south of this South Reef.

I submit that there is every evidence to show that this dyke has duplicated the Main Reef Series in the East Rand district. The North and South Series were shown in the previous chapter each to include the South Reef, Main Reef Leader and Main Reef of the Main Reef Series.

The North Series has throughout a lower dip than the South Series, as shown in the following table :—

	Dip of North Reef.	Dip of South Reef.
Driefontein . . .	45°	55°
Angelo . . .	45°	55°
New Comet . .	47°	57°

At these two angles of dip the reefs are approaching in depth, as is shown in the New Comet, where at the surface the reefs are 550 feet apart, whereas in the main cross-cut on the 5th level, which passes through the dyke, this distance is under 500 feet. Further, the termination of the South Reef against the dyke has been traced from the New Comet mine to the property of the Witwatersrand Deep.

Granted then the duplication of the Series, the upthrow on the south side must have been about 600 feet. The course of this dyke to the west is such that it is likely to have caused the upthrow which has been experienced in the Lohse shaft of the Simmer East. In that shaft, which is 5080 feet from the outcrop, the Main Reef Series was struck at a depth of 1863 feet by a diamond drill, the core showing an apparent dip of 40°; whereas in No. 1 shaft of the Rose Deep, which is 1820 feet from the outcrop and in line with the Lohse shaft, the reef was struck at a depth of 942 feet.

Balmoral Longitudinal Dyke.—This dyke intercepts the South Series as it comes from the Witwatersrand property, at a distance of about 700 feet from the western boundary, and causes a duplication on the surface for about 570 feet, which is equivalent to a considerable upthrow on the south side. It strikes about E.N.E. and has a high angle of dip.

Farther to the east, in the Van Ryn-Chimes district, the Main Reef Series was struck in a bore-hole situated 4340 feet from the outcrop on the property of the Chimes mines, at a depth of 1738 feet. The estimated depth at the average angle of dip which obtains in this district was much less, the difference being due either to greater flattening of dip or to an upthrow, the latter being the more likely.

Village Main Reef Fault.—This fault, which gave a good

deal of trouble in developing the mine, can be traced to extend on the west side till it crosses the outcrop of the Main Reef Leader in the Wemmer, and on the east side through the City and Suburban, Meyer and Charlton, and Wolhuter mines, till it recrosses the outcrop. The distance between the two places where it crosses the outcrop is about 1¾ miles. Along this length it dips about 70° to the south, strikes roughly parallel with the formation, and is accompanied by a downthrow of about 30 feet, so that it is a normal fault.

These are the principal longitudinal dykes and faults along the Rand. With few exceptions, they are in strike, dip, and throw so similar as to suggest some common origin.

It is seen from the tabulated statement on p. 120 that they dip much more frequently to the south than to the north, and that the average angle of dip of those occurring in the Central Rand is 69°, an angle considerably lower than that of the transverse dykes.

There is another feature exhibited longitudinally along the formation, namely, a secondary steepening of the reefs in depth, which has already been described.

The connection between these longitudinal undulations and the longitudinal reverse faults would appear to be very close. Some of the small reverse faults gradually lose their intensity till a bend in the strata takes the place of the fracture, and this continuing in the same direction eventually dies out.

The flattening which can be followed from the Simmer and Jack into the Geldenhuis Estate is found in the latter mine accompanied by small reverse faults; and in the Primrose mine, where, in the central section, a flattening occurs between the 4th and 5th levels, there is a fracture dipping south, and accompanied by reverse faulting, which in one place breaks the continuity of the upper reef, but only bends the lower one.

In view of these facts the longitudinal undulations may be

regarded as marking an early stage in the formation of the east and west reverse faults.

This direction east and west is parallel to the axis of the granite lying to the north, and it would appear that these fractures and undulations were developed at the formation of the Witwatersrand Syncline by the granite, either by direct action exerted at its upheaval, or by reaction against lateral thrusts from its surface. This conception of their formation assigns to the occurrence of these faults a stability above that of any other series of faults yet met with, or likely to be found in the whole district.

It is probable that the upthrows on the south side, which we now find associated with them, occurred at some later date, because the former have not as yet shown any decrease in intensity receding from the granite.

Across the formation from the Glencairn to the Lohse shaft of the Simmer East there are at least two of these reverse faults, of which the one farther from the outcrop is the greater. The first, met with in the ground of the Glencairn itself, is a continuation of the Simmer Dyke, and it has an upthrow on the south side of about 400 feet. The second occurs somewhere between the Glen Deep No. 1 shaft and the Lohse shaft, and its intensity is estimated as an upthrow of about 1000 feet on the south side.

Across the formation at the Crown Reef, between the Main Reef Series and the Kimberley Reef, there are at least three of these reverse faults, viz. :—

The Ferreira-Crown Deep Dyke,
The Grahamstown Dyke, and
The South Rand Dyke.

Of these the first has an upthrow of 75 feet on the south side, the second of 400 feet, and the third of 270 feet.

In view of the increase in value given to some areas by this reverse faulting, it is worth while considering to what extent it is likely to recur.

The effect of longitudinal reverse faulting occurring in a formation is to increase the width of its outcrop on the surface. By a series of such reverse faults one bed might be brought up to the surface so many times so as to give the appearance of great width on the surface. There is no doubt that the great apparent thickness of the Witwatersrand beds south of the Rand is due in great measure to such reverse faults, some of which have been discovered, others indicated, leaving most probably some still absolutely undiscovered.

Bedded Dislocations.—The most noticeable of these is that which occurs in the Central Rand between the Main Reef and Main Reef Leader. This last reef is generally characterised by its strong attachment to the quartzite in the hanging wall, into which it is often described as being "frozen," and by the occurrence of a strong and distinct parting along its footwall. This footwall parting would appear to be a separation subsequent to its deposition, because in many places it is too strongly developed to be due to bedding, and, while it occurs almost invariably in the Central Rand, it is not distinguishable in the mines west of Langlaagte. Near Johannesburg it is marked by a characteristic occurrence of soft schists, accompanied in many places by a vein of secondary quartz. There are areas in the Ferreira, Robinson, and Robinson Deep mines where it does not appear, and it is the experience in the Robinson mine that over such areas the Main Reef Leader is not so good as elsewhere.

Going east from Johannesburg this parting becomes more marked; in places it bellies out to considerable thicknesses as a compact rock, in place of the soft schists. Such an occurrence is found below the 9th level in the City and Suburban and in the eastern portion of the Wolhuter.

In the Spes Bona this intrusion has been keenly felt. In this mine it comes in between the Main Reef Leader

and Main Reef just below the 3rd level, as shown in the following section (Fig. 33) through the main shaft of that mine.

On the 3rd level the two reefs are separated by about 8 feet of quartzite, whereas on the 4th level there is a horizontal thickness of 40 feet between them, this distance being occupied by a rock looking like a hard siliceous mudstone. The upper reef seems to have been enriched by this intrusion, but the Main Reef is considerably brecciated and rendered valueless. This intrusion continues with varying

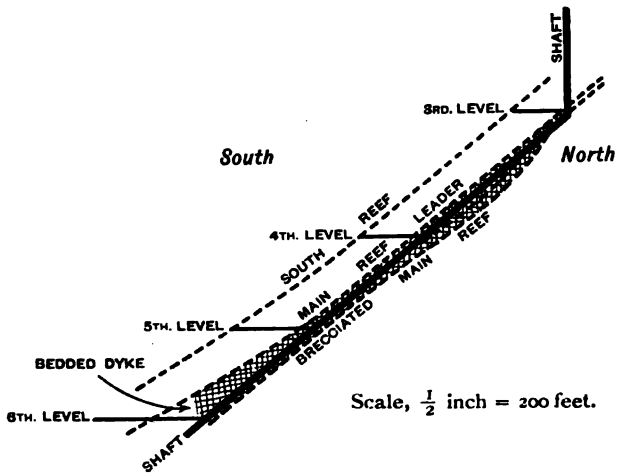


FIG. 33.—Section through the Main Shaft of the Spes Bona, showing the occurrence of a bedded dyke.

thickness as far as sinking has been carried as yet in this mine, but in the deep-level shaft of the Wolhuter, which is immediately to the south, it has practically disappeared. In the Geldenhuis Estate the occurrence of a band of schist under the Main Reef Leader has characterised that reef, so that it is often spoken of as the "Slate Leader."

In the Simmer and Jack the Main Reef Leader, called there the Middle Reef, maintaining its characteristics of being frozen into the hanging wall and free along its footwall, has a thickness of this dyke matter underneath it, which is generally accompanied by veins of quartz.

In the New Primrose mine this schist has long been recognised as an altered igneous rock; on it the Main Reef Leader, called here the Middle Reef, lies. Cross sections through this mine show the nature of this occurrence extremely well. It varies greatly in thickness, presenting an irregular section in every direction; sometimes it separates the Middle and North Reef for 160 feet, as in the western portion of the mine; and in places it thins out till the clay and contact quartz alone are left. Wherever it is thick the underlying reef is broken up considerably, so that it becomes in places too poor to work.

In the May Consolidated it is found in position under the slate leader, and in places it has, enclosed within its mass, small pebbles of quartz, giving it somewhat the appearance of a poor banket, so that it is known in this mine as the "bastard reef."

In the Loshe shaft of the Simmer East, at a distance of 5080 feet from the outcrop, and at a depth of 1849 feet the Main Reef Leader was struck in a bore-hole, having under it a bed of shale several inches thick.

It is represented by a strong parting under the Middle Reef of the Glencairn, by the Bastard Reef under the Leader of the South Reef in the Witwatersrand mine, and throughout the mines of the East Rand Proprietary it is the strong parting under the richest portion of the reef, known in the New Blue Sky as the slate footwall. In the Van Ryn-Chimes district it is not noticeable.

This plane of bedding, accentuated by the intrusion of dyke matter, is in the Central Rand the best guide for the correlation of the several members of the Main Reef Series.

In the Worcester mine there is an interesting occurrence of a bedded dyke, about 4 feet thick, which from the surface down to between the 5th and 6th levels lies just in the hanging wall of the Main Reef Leader. There it crosses this reef and forms its footwall in all the stopes below. The

line along which the transference from the hanging wall to the footwall takes place runs more or less horizontally right into the Robinson on the west side; on the east, when near the Ferreira, this dyke crosses the Main Reef Leader again, and strikes out into the hanging wall. The dyke rock is gray, compact and hard, looking like a siliceous mudstone.

It was noticeable that in the upper levels, where this dyke was above the Main Reef Leader, this reef was richer in its hanging wall; in the lower levels, where it is below the reef, the footwall of the reef is richer. The underlying Main Reef, so far as it has been opened up, does not appear to have been disturbed. The occurrence is illustrated in the following sketch (Fig. 34).

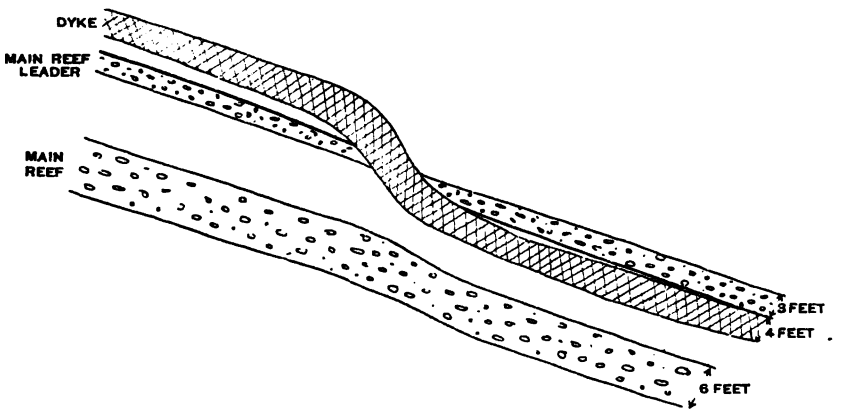


FIG. 34.—Occurrence of an interbedded dyke between the 5th and 6th levels of the Worcester.

In the Ferreira mine there is a similar occurrence of a dyke cutting across the formation in one part of its extent and following the bedding in another. This is the small dyke west of No. 1 shaft, which, coming from the south, cuts across the formation in a north-east direction till it reaches the Main Reef Leader, when it turns to follow the footwall of that reef. It seems likely that this occurrence is coincident with that just mentioned in the Worcester, and that these two form part of the ramifications of the same intrusion.

Along the 820 foot level on the South Reef, west of

No. 1 shaft of the same mine, there was another limited extent of bedded igneous rock, which was rendered most interesting by the occurrence of a good deal of gold within its mass. It extended for about 30 feet along the level and 50 feet above, up to meet the large Ferreira dyke. It was found lying underneath the South Reef, and separated from it by a quartz vein. The occurrence is shown in the sketch below (Fig. 35).

It was about 8 inches thick up against the large dyke. Descending to the level its size diminished till, at the level,

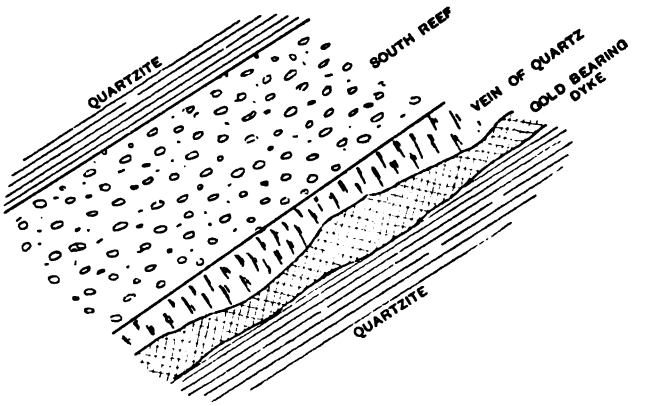


FIG. 35.—Occurrence of a gold-bearing bedded dyke in the Ferreira. 820 foot level; west of No. 1 shaft.

it had thinned out entirely. It does not occur on the north side of the dyke, nor in any other part of the mine. The dyke rock was dark, soft, and without schistose structure.

There is a strong bedded dyke overlying the Bird Reef Series in the Central Rand. From a position south of the Pioneer its outcrop can be traced on to the Robinson Deep ground, where it passes underneath a depth of surface. Along this stretch it is about 60 feet wide, and it dips more or less conformably with the strata; its distance south of the Bird Reef Series varies from 80 to 220 feet. This same dyke outcrops again on the Jupiter property, and can be traced past Elandsfontein Station. It has a characteristic dull red colour on its surface, but broken pieces show a green

or gray tinge inside. It is very hard and compact, and it is jointed along planes parallel to the bedding.

Along the Central Rand it keeps its position above the Bird Reef so well, that it has been used as a guide to the depth of the Main Reef below. It was the occurrence of this dyke south of the Lohse shaft of the Simmer East that first suggested that the Main Reef Series might not be so deep in this shaft as the distance from the outcrop would lead one to expect.

A little to the north of the Great Britain mine, which lies south of Roodepoort, a similar dyke occurs, and farther to the west, in the Krugersdorp district, one occurs running along the formation about 200 feet north of the battery Reef in the Violet mine.

In the Klerksdorp district bedded dykes are shown by Denny [1] in sections across the formation at Buffelsdoorn, Elandsheuvel, and Rietkuil. In any one of these sections the dykes appear perfectly interstratified, but traced along the surface from one point to another, they are seen in places—notably on the farm Elandsheuvel—to break across and greatly disturb the formation.

They occur along the horizon of the larger-pebbled banket beds, in some cases being found in contact with them, as on the farms Rietkuil and Buffelsdoorn. In this district, however, these dykes are not in superficial extent comparable to the more recent diabasic overflows which have occurred along fissures more or less parallel with the formation. It is likely that some of the shale beds of the Quartzite-Shale group, which underlies the Witwatersrand Beds, are but the weathered outcrops of igneous rocks.

The comparative age of a dyke can be determined by its intersections with other dykes.

One of the oldest dykes near Johannesburg is the Salisbury dyke, No. 24 in the statement on p. 118. This dyke

[1] *The Klerksdorp Goldfields*, by G. A. Denny.

crosses the outcrop of the Main Reef series in the property of the Salisbury, but it has also been found in the Wemmer and Ferreira.

In the Wemmer mine it is thrown by all the dykes which cross it, as shown in the following plan (Fig. 36) of that property :—

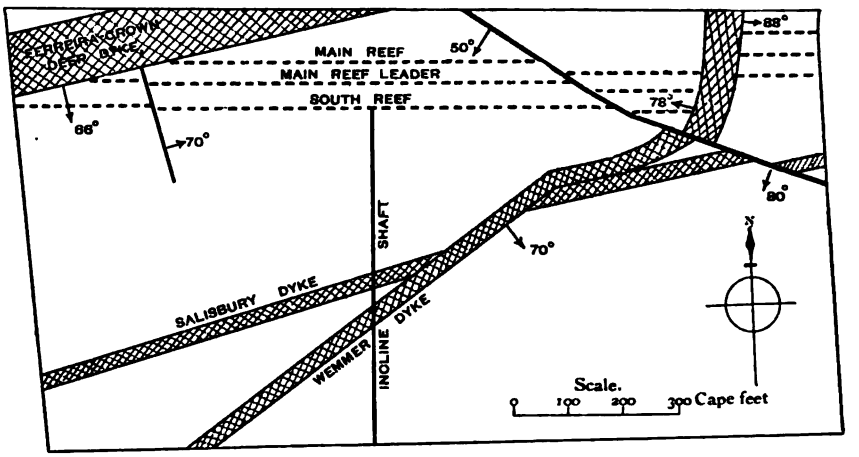


FIG. 36.—Plan of the outcrop portion of the Wemmer, showing the intersection of dykes, faults, and reefs with the plane of the surface.

Similarly, in the Ferreira its course is very irregular, owing to the amount of displacement it has suffered from

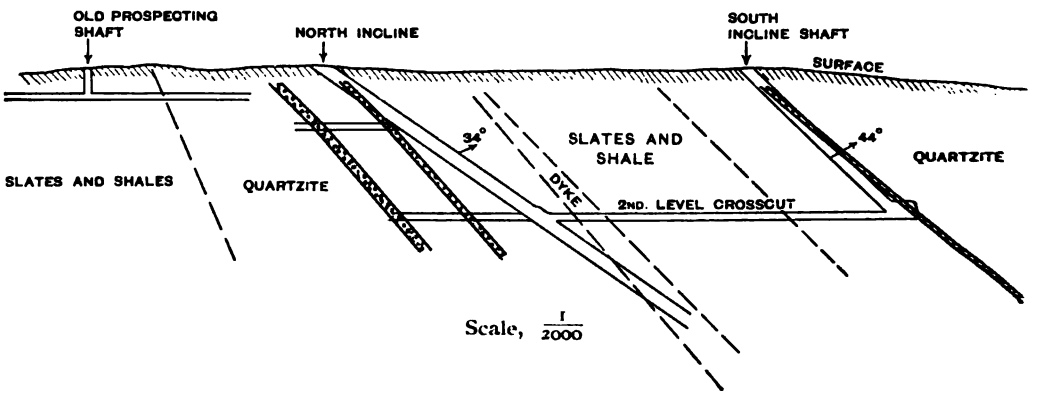


FIG. 37.—New Blue Sky, East Section, cross section showing reefs and dyke matter.

other dykes. Its intersections with the Ferreira-Crown Deep dyke are shown on p. 116, Fig. 38.

This dyke is about 27 feet thick, dips to the north, and by its course is classed as longitudinal. The rock has a dull colour, with no appearance of crystalline structure. The longitudinal dykes generally, by their intersections, appear to be of earlier date than the others, and of them, those that dip to the north appear to be the oldest. It would be interesting if this last fact could be substantiated, because it might follow that they were intruded before the beds had been tilted, and might have been given their dip to the north by the swinging of the beds in tilting.

Referring to the tabulated results on p. 120, the following points deserve notice. That of the forty-six dislocations in the Central Rand:—

(1) There are twice as many transverse as longitudinal dislocations.

(2) There are twice as many dislocations which dip to the south as dip to the north.

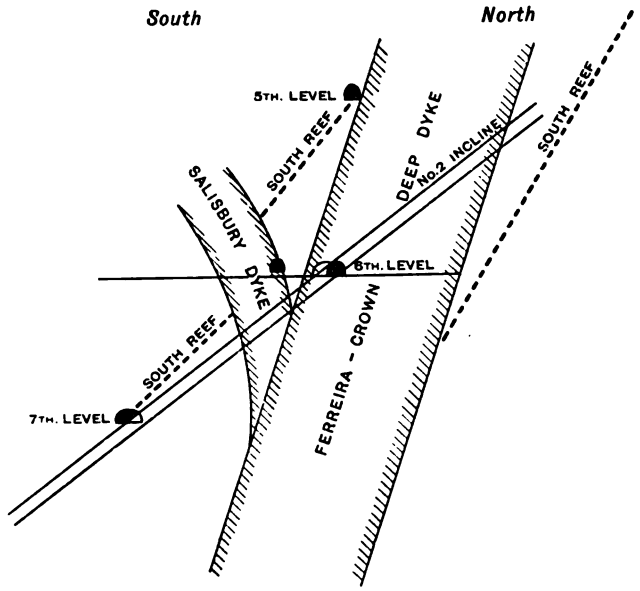
(3) The transverse dislocations dip at a steeper angle than the longitudinal.

(4) The longitudinal dislocations are almost all right-handed, indicating a series.

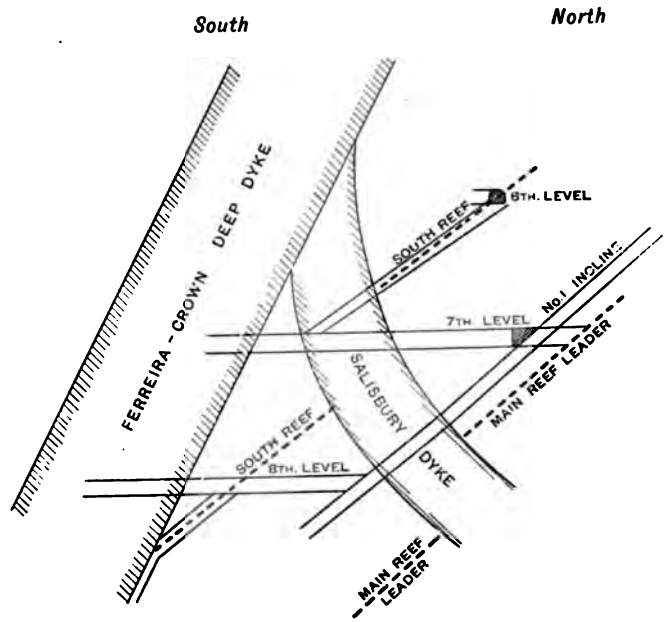
(5) That of the total, forty-six, only seven are reverse dislocations, and that

(6) Of these seven, six are longitudinal dislocations, and the remaining one is a transverse dislocation.

Out of these dislocations, twenty-three, or one-half, were dykes having an average thickness of about 50 feet, whilst the other half are classed as faults, their thickness being inconsiderable.



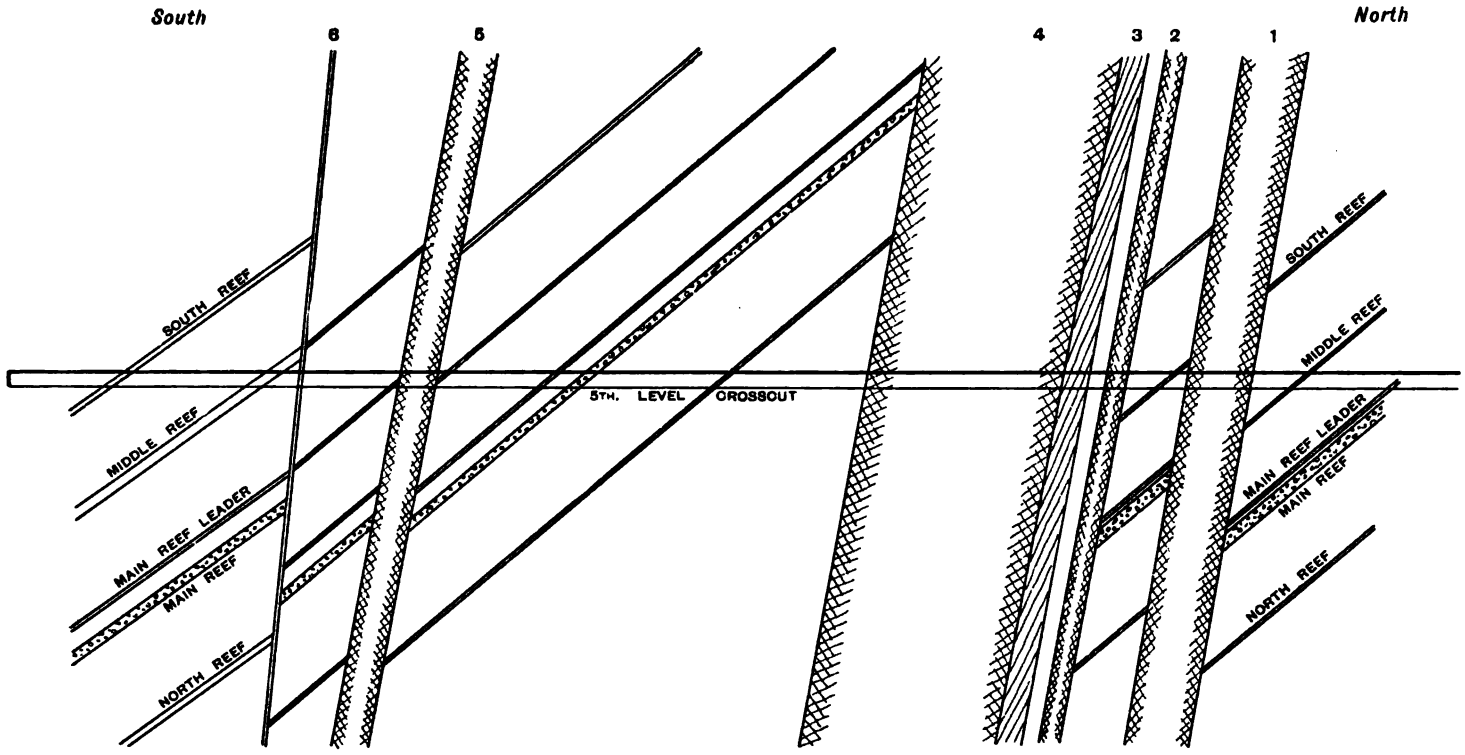
SECTION THROUGH No.2 SHAFT



SECTION THROUGH No.1 SHAFT

Scale, $\frac{1}{1000}$

FIG. 38.—Sections through the two shafts of the Ferreira mine showing the upper and lower intersections of the Ferreira-Crown Deep dyke with the Salisbury dyke.



Scale, 1 inch = 100 feet.

Dykes.—No. 1, fine-grained, medium hardness.
No. 2, ,, ,, ,,
No. 3, schistose structure.
No. 4, coarse-grained, very hard.
No. 5, ,, decomposed.
No. 6, fine-grained, schistose, medium hardness.

FIG. 39.—Section of the formation afforded by the south cross-cut, 5th level, Langlaagte United.

DYKES AND FAULTS CROSSING THE OUTCROP OF THE MAIN REEF SERIES FROM THE CROWN REEF TO THE GEORGE GOCH

Name of Company.		Position or Name.	Thickness.	Strike.		Dip.		Throw in Feet.				
			Feet.	Direction.	Degrees.	Direction.	Degrees.	N.	S.	Right or Left.	Normal or Reverse.	Transverse or Long.
Crown Reef .	1	Fault west of Main Incline .	...	N. by E.	5	E. by S.	89	58	...	Left	Normal	Trans.
	2	Fault east ,, .	...	S.S.E.	166	...	90	...	57	Right	,,	,,
	3	Fault near No. 1 Shaft .	...	N.N.E.	8	...	90	17	...	Left	,,	,,
Robinson .	4	No. 1 numbered from the west	17	S.E.	126	S.W.	65	...	35	Right	,,	Long.
	5	,, 2 ,, ,,	25	S.S.E.	168	S.W.	70	...	40	,,	,,	Trans.
	6	,, 3 ,, ,, (Crown Reef dyke) .	12	E.N.E.	75	N.N.W.	85	...	60	,,	Reverse	Long.
	7	,, 4 large Robinson dyke .	230	N.	1	W. by N.	70	...	None	...	...	Trans.
	8	,, 5 . . .	8	S.E.	131	S.W.	56	...	Small throw	Right	Normal	Long.
	9	,, 6 longitudinal dyke .	30	E.N.E.	62	S.S.E.	54	...	2	,,	Reverse	,,
	10	,, 7 (met with in Crown Deep) . .	10	S.E.	145	S.W.	75	...	10	,,	Normal	Trans.
	11	,, 8 (met with in Robinson Deep) . .	38	E.S.E.	102	S.W.	55	15	...	Left	,,	Long.
	12	,, 9 . . .	20	S.S.E.	171	W.S.W.	85	...	20	Right	,,	Trans.
	13	,, 10 . . .	15	S.E.	136	S.W.	74	None	None	...	...	,,
	14	,, 11 . . .	...	S. by E.	175	W. by S.	70	...	87	Right	Normal	,,
	15	,, 12 . . .	...	N.E.	35	S.E.	78	62	...	Left	,,	,,
Worcester .	16	Main dyke . . .	21	S.S.E.	166	W.S.W.	80	...	30	Right	,,	,,
	17	Fault east of shaft . .	...	S.E.	141	N.E.	86	18	...	Left	,,	,,
Ferreira .	18	Small dyke west of No. 1 Shaft	4	S.E.	133	S.W.	64	...	34	Right	,,	Long.
	19	Large dyke . . .	125	E.N.E.	80	S.S.W.	65	...	20	,,	Reverse	,,
	20	Overlap fault . .	...	N.E.	51	N.W.	83	...	20	,,	,,	,,
Wemmer .	21	Fault 400 feet west of shaft .	...	S.S.E.	156	E.N.E.	67	...	8	,,	,,	Trans.
	22	Small dyke 400 feet east of shaft, Village Main Reef fault	4	E.S.E.	122	S.S.W.	77	...	20	,,	Normal	Long.

	23	Dyke near eastern boundary .	30	N.E.	38	S.E.	70	78	...	Left	Normal	Trans.
Salisbury .	24	Longitudinal dyke from Wemmer . . .	27	E.N.E.	80	N.N.W.	74	...	4	Right	"	Long.
	25	East fault . . .	...	N.N.E.	29	E.S.E.	77	33	...	Left	"	Trans.
Jubilee .	26	Grahamstown dyke . .	230	N.N.E.	20	E.S.E.	86	...	300	Right	Reverse	Long.
City and Suburban .	27	No. 1 from west boundary .	...	S.S.E.	170	E.N.E.	88	37	...	Left	Normal	Trans.
	28	" 2 . . .	...	N.N.E.	22	W.N.W.	80	17	...	"	"	"
	29	" 3 . . .	...	N.N.E.	18	E.S.E.	85	19	...	"	"	"
	30	" 4 . . .	...	S.E.	147	S.W.	80	...	26	Right	"	"
	31	" 5 . . .	...	S. by E.	177	E. by N.	85	25	...	Left	"	"
Meyer and Charlton .	32	" 1 . . .	...	S.S.E.	152	W.S.W.	80	...	20	Right	"	"
	33	" 2 . . .	...	N.N.E.	9	W.N.W.	85	26	...	Left	"	"
	34	" 3 . . .	...	N.E.	31	S.E.	77	12	...	"	"	"
	35	" 4 . . .	...	N.N.E.	11	E.S.E.	72	47	...	"	"	"
	36	" 5 . . .	...	S.S.E.	151	W.S.W.	55	...	10	Right	"	"
	37	Large dyke . . .	94	N.E.	31	S.E.	89	80	...	Left	"	"
	38	West dyke . . .	45	N.N.E.	20	...	90	...	27	Right	"	"
	39	Fault near Main Incline .	...	N.N.E.	22	...	90	20	...	Left	"	"
	40	Eastern fault . .	...	N.N.E.	22	...	90	20	...	"	"	"
Spes Bona .	41	Fault west of Main Shaft .	...	N. by E.	4	W. by N.	57	40	...	"	"	"
	42	Spes Bona large fault .	8	S.E.	135	S.W.	50	...	312	Right	"	Long.
George Goch	43	George Goch large fault .	8	E.S.E.	110	N.N.E.	80	400	...	Left	Reverse	"
	44	Dyke in No. 2 Mine west of shaft . . .	110	N.N.E.	22	...	90	None	None	...	...	Trans.
	45	Central fault . .	...	E.S.E.	110	S.S.W.	72	...	50	Right	Normal	Long.
	46	Metropolitan fault . .	20	E.S.E.	120	N.N.E.	70	...	480	"	"	"
								1024	1672			

TABULATED RESULTS FROM THE FOREGOING STATEMENT OF DYKES AND FAULTS

Distance covered . . .	5 miles
Resultant throw to the south .	648 feet
Average strike of formation .	east and west

Classification according to Strike

Transverse	31
Longitudinal	15
	46

Classification according to Direction of Dip

	Trans.	Long.	Total.
Dip south . .	17	10	27
„ north . .	8	5	13
Vertical . .	6	0	6
Totals .	31	15	46
Dip west . .	13	11	24
Dip east . .	12	4	16
Vertical . .	6	0	6
Totals .	31	15	46
Average dip .	83°	69°	78°

Classification according to Nature of Throw

Right-handed . .	11	13	24
Left-handed . .	17	2	19
No throw . .	3	0	3
	31	15	46
Normal . .	27	9	36
Reverse . .	1	6	7
No throw . .	3	0	3
	31	15	46

SERIES OF FAULTS AND DYKES IN THE VAN RYN WEST AND VAN RYN MINES DESCRIBED IN ORDER FROM WEST TO EAST

No.	Dyke or Fault.	Thickness.	Approximate Position of the Western Termination of the Main Reef against the Dislocation.	Strike.	Dip.	Amount of Throw measured across the Formation.	Nature of the Throw.		Amount and Direction of the Alteration in Strike of the Reefs.
							Right or Left Hand.	Normal or Reverse.	
1	Dyke	...	Van Ryn West, 1500 ft. from east boundary . . .	N.N.W. and S.S.E.	...	300 ft. S.	Right	Normal	20° S.
2	"	...	Van Ryn West to Van Ryn, 350 ft. from east boundary .	W.N.W. and E.S.E.	...	200 ft. "	"	"	6½° N.
3	"	84 ft.	Van Ryn, 1200 ft. from west boundary	N. and S.	...	700 ft. "	"	Reverse	25° N.
4	Fault	...	Van Ryn, 170 ft. west of No. 4 Shaft	W.N.W. and E.S.E.	60° S.	30 ft. "	"	Normal	
5	"	...	Van Ryn, 30 ft. west of No. 1 Shaft	W. 15° N. and E. 15° S.	70° S.	16 ft. "	"	"	
6	"	...	Van Ryn, 120 ft. N.E. of No. 1 Shaft	W.N.W. and E.S.E.	70° S.	16 ft. "	"	"	7½° S.
7	"	...	Van Ryn, 680 ft. east of No. 1 Shaft	W. 30° N. and E. 30° S.	70° S.	20 ft. "	"	"	
8	"	...	Van Ryn, 1100 ft. east of No. 1 Shaft	E. and W.	Vertical	16 ft. "	"	"	
9	"	...	Van Ryn, 1180 ft. east of No. 1 Shaft	W.S.W. and E.N.E.	85° S.	considerable S.	"	"	10° S.
				Resultant throw S. . .		1298 ft.			6° S.

Resultant change of strike of the reefs in the direction of the throw, 6° S.

All these dislocations can be traced to cross the outcrop of the Chimes Reef. The resultant alteration of the strike over this extent is only 6° S., but the difference between the strike of the reef in the New Kleinfontein and that in the Modderfontein Extension is about 63° S., and these two properties occur, the one immediately to the west of No. 1 dyke, and the other to the east of No. 9.

SERIES OF FAULTS OCCURRING IN THE WEST RAND ON THE PROPERTY OF THE LANCASTER, GOING FROM EAST TO WEST

No.	Name of Reef Series.	Approximate Position of the Eastern Termination of the Reef against the Fault.	Strike of Fault.	Angle of Increase of Reef Strike to the South.	Amount of Throw measured across the Formation.		Nature of Throw.	
					South.	North.	Right or Left Handed.	Normal or Reverse.
					Feet.	Feet.		
1	Keeley's Reef	Lancaster, 500 ft. from western boundary	N.E. and S.W.	4°	95	...	Left	Normal
2	Monarch	,, 400 ,, eastern ,,	,,	3½°	135	...	,,	,,
3	,,	,, 835 ,, ,, ,,	N.N.E. and S.S.W.	6½°	...	75	Right	Reverse
4	,,	,, 1540 ,, ,, ,,	N.E. and S.W.	21°	45	...	Left	Normal
5	,,	,, 2535 ,, ,, ,,	,,	5°	245	...	,,	,,
6	West Battery	,, 335 ,, ,, ,,	,,	21°	17	...	,,	,,
7	Reef Battery	,, 735 ,, ,, ,,	,,	(−2°)	210	...	,,	,,
	,,	,, 1170 ,, ,, ,,	N.N.W. and S.S.E.	0°	...	9	Right	,,
			Totals . .	61°	747	84		
				2°	84			
			Resultants .	59°	663			

The distance along the formation between the extreme east and west faults is one mile.
Resultant throw south by faulting, 663 feet.
Angle through which the strike has turned farther south, 59°.

CHAPTER VI

THE GENESIS OF THE WITWATERSRAND BANKET

MR. TRUSCOTT has discussed the geotectonic geology of the district, and I shall therefore confine this chapter to a consideration of the theories relative to the ore genesis of the auriferous beds. Several theories have been advanced to explain the origin of the conglomerate beds, but the only one deserving serious consideration is that they are littoral sedimentaries. Neither the fluviatile nor the lacustrine hypotheses square with the observed phenomena. The consensus of opinion of geologists who have investigated the subject is that the beds of sandstone and quartzite are the metamorphosed accumulations of marginal sea-shore deposits formed during periods of subsidence of the coast-line. In the process of the earth's evolution, through causes known to students of dynamical geology, the proximate cause of which was undoubtedly the intrusion of a large mass of granitic rocks from the north, the banket-bearing strata were folded into an anticline and syncline. Subsequent erosion has completely removed the anticline. The metamorphism of the arenaceous shore-deposits into highly indurated sandstones and quartzites accompanied the upheaval. There exists, however, a diversity of opinion regarding the origin of the gold in the conglomerate beds. The theory of its fluviatile origin has the professional endorsement of only a few engineers.

The precipitation theory recently advanced by Monsieur

de Launay[1] is an elaboration of a theory previously suggested by Mr. Penning. This theory maintains that the sea, in which the banket was deposited, was a saturated solution of gold and pyrite, and that the auriferous contents were deposited *pari passu* with the accumulation of the conglomerate pebbles.

A valid objection to this theory is that the gold is confined to the strata of pebbles, *i.e.* the banket, and does not occur in the interstratified sedimentary rocks *except rarely where such rocks occur as "horses" or inclusions in the conglomerate bed.*

In a recent publication by Don[2] he states that all his experiments have signally failed to show any precipitation of gold from sea-water by natural reagents, and, notwithstanding most careful and extensive investigations, he finds no support to the theory that the deposition of gold and silver by such reagents in marine sediments is now going on. He made a careful examination of many stratified rocks known to be consolidated marine sediments, and in no instance detected in these rocks the presence of gold which could be ascribed to precipitation during the period of sedimentation, and he asks the question, which is pertinent to the present discussion, "If such deposition had been the rule in former periods, and that this be the origin of the gold in stratified formations, why should only a comparatively small portion of such formation be traversed by auriferous veins?"

According to De Launay's theory, there must either have been an intermittency in the action of the precipitation of gold from its menstruum, or there must have been a periodicity in the occurrence of the saturated solution of gold and pyrite in the sea to explain the gold tenure of the gravel

[1] *Les Mines d'or du Transvaal.* Published by Baudray and Co., Paris.

[2] "The Genesis of Certain Auriferous Lodes," *Transactions of the American Institute of Mining Engineers.*

beds and its total absence in the interstratified sandstones and quartzites. Either hypothesis seems untenable.

The marine theory of Dr. Becker[1] refers the gold contents of the reef to placer origin, ascribing, however, the presence of coarse gold to secondary action. According to this theory the bulk of the gold should invariably occur in the footwall, or lowest portions at least, of the reefs, as the result of the concentration to which the pebbles composing the conglomerate beds were subjected by the action of tidal waves during their deposition. The larger pebbles generally occur near the footwall of the reef, and the bulk of the gold, it is true, usually occurs with the coarser pebbles, but in some mines the hanging wall portions of the reef, notwithstanding the fact that the pebbles are smaller, contain the bulk of the gold, and more rarely the central portion of the reef is found to be the richest. Many of the larger-pebbled conglomerate beds situated near, and running parallel to, the Main Reef Series contain but traces of gold. Indeed, in the Main Reef Series itself, the South Reef, which contains the smallest pebbles of the associated bankets, is almost invariably the richest. There are several other important facts which militate against this theory, among which is the occurrence of well-defined pay-shoots in the reef. Dr. Becker admits the force of this objection, but questions the existence of such shoots of ore, and states that their occurrence would involve the assumption that there are two classes of mineralised reefs *i.e.* that the reefs where these ore-shoots are found were mineralised according to the third theory, which he describes as the impregnation theory, to which I shall presently refer. There are most unquestionable evidences of the existence of such shoots of ore in the Nigel and Rietfontein mines. There are also evidences of shoots on other portions of the Rand, but I agree with Dr. Becker that in the majority

[1] *Eighteenth Annual Report of the United States Geological Survey*, 1896-97, part v. p. 153.

of cases the payable ore occurs in patches rather than in defined shoots. Indeed, it is sometimes difficult to distinguish between a pay-shoot and a mineralised patch. The elongated patches sometimes partake of the shape of ore-shoots, but, on the other hand, pay-ore-shoots in quartz veins frequently assume shapes identical with those of many of the so-called pay-ore patches of the Rand. The line of demarcation is at times obscure.

The impregnation theory suggested by Messrs. Gardner, F. Williams, and J. S. Curtis commends itself most favourably to the resident mining engineers of the district. This theory attributes the presence of gold and pyrite in the reefs to deposition from infiltrating solutions, as in the genesis of auriferous quartz veins. According to this view, the mineralising solutions have come up along the planes of least resistance, *i.e.* the interstitial spaces of the banket reefs, and the gold therefore has been deposited since the upheaval of the conglomerate beds. Some specious objections have been urged against this theory, amongst others, the fact that there is an absence of fissures in some localities where the ore is nevertheless of good grade. Such an objection would apply with equal force to the theory generally of the mineralisation of quartz reefs by ascending solutions, for in such reefs there are frequently long stretches of payable ore without any recognisable connection with such fissures. The gold-bearing solutions came up, as I have stated, along the planes of bedding of the banket, and these planes of bedding have undoubtedly had connection with fissures, though generally deep-seated, and therefore not always visible.

Dr. Becker states that there is no connection whatever between the tenure of the banket and the numerous dykes which traverse the formation. The statement is generally true that there is as yet no ascertained connection between the richness of the bankets and these dykes. There are, however, some most important exceptions to this assertion.

Indeed, numerous examples could be adduced proving local enrichment of the reefs due to proximity of dykes. A very instructive illustration is in the Worcester mine (see p. 111), where there is an intrusive dyke lying in the upper levels, immediately above the Main Reef Leader. The hanging wall is very much richer than the footwall portion of the reef, and maintains this character until the lower levels are reached, when the dyke having crossed the banket, forms its footwall. From this point the bulk of the gold contents of the reef is found on the footwall. Numerous other examples could be adduced, but it will suffice to instance the Buffelsdoorn mine, where the payable section of the reef is determined entirely by the proximity of the dykes.

In the Ferreira mine there is an interesting occurrence of a dyke containing a considerable amount of visible gold. This dyke lies below the South Reef, and is separated from it by several inches of vein quartz, as shown on p. 112. The quartz itself is distinctly auriferous. Associated with the banket beds, sometimes following and crossing the formation, at other times intercalated with the banket, are numerous quartz veins. While these quartz veins usually carry but little gold, they have, however, in places been found very rich, and have yielded considerable gold when crushed in the battery. As the result of the movement connected with the upheaval of the formation many of the quartz pebbles were cracked, but have been re-cemented by infiltrating quartz solutions. Upon the re-cemented cracks are frequently found gold lamellæ. Sometimes the quartz pebbles are found encrusted by minute crystals of quartz, which must have been deposited from infiltrating siliceous solutions after the upheaval of the beds. Combs and geodes, similar to those found in quartz veins, have been found in the reef.

The resemblance between the fineness of the gold in the banket and that of the accompanying quartz reefs has, in a

few instances, been proved, though investigations have been too limited to establish a general rule. Coarse gold is frequently found not only in the vadose regions of the reef, but in the deeper portions as well.

The foregoing facts point unmistakably to the mineralisation of the reefs in part, according to the impregnation theory. The reefs contain, however, a considerable amount of gold of undoubted detrital origin, as proved by its water-worn appearance. This one would expect as the result from the origin of the banket beds themselves, the quartz pebbles of which have been undoubtedly derived from the degradation of pre-existing quartz reefs.

JOHN HAYS HAMMOND.

CHAPTER VII

PROSPECTING

Topography.—The topographic features of the country are entirely due to the geological formation.

The granite, which forms the outside envelope of the beds, occupies an encircling depression, within which more recent measures have often been deposited.

The quartzites of the Quartzite-Shale group have resisted disintegration, so that this series generally occupies the highlands on which the course of the interbedded shales is marked by depressions running parallel to the formation; it is on the inside slope of this group that the true banket formation or Witwatersrand Beds occur.

Where the amygdaloidal diabase is largely developed, it stands high, having resisted the denuding agencies, while the dolomite formation generally lies low; lastly, the Gatsrand quartzites form a range of higher ground which runs axially down the length of the syncline.

Generally the hill ranges and the depressions follow the contour of the syncline.

The courses of the larger dykes and dislocations are often impressed on the surface; thus along the Witpoortje Break there runs a stream. The Robinson Spruit (*i.e.* stream) runs close to and parallel with the large Robinson dyke till the larger South Rand dyke is met, when it changes its direction to follow the latter dyke along the southern boundary of the Langlaagte farm. The Metropolitan fault has caused a

depression on the surface, and the course of the spruit between the eastern and western portions of the Blue Sky mine probably marks the occurrence of a dyke ; the property of the Witwatersrand Gold Mining Company is intersected on the surface by spruits which prospecting work has proved to be due to the effects of dykes and disturbances. Where the banket formation is only covered by a small thickness of subsoil, the ant-hills on the surface are generally of a yellowish-gray colour ; dykes near the surface are generally decomposed into a soft and homogeneous red loam, so that, should such a one cross the formation, its course can be followed by the red colour of the ant-hills.

General Considerations.—The aim of prospecting is to determine the occurrence and value of reefs.

For the banket formation the greatest aid in prospecting is a correct knowledge of the sequence of the strata ; this is to be determined by making a geological section across the formation ; the position in this section, occupied by the ground being prospected, has a great bearing upon the likelihood of finding valuable reefs in it.

The various reefs are well known to occur at distinct horizons, outside of which they are not to be found ; thus the Main Reef occurs near the base of the Witwatersrand Beds and close to the Quartzite-Shale group, and the Bird, Kimberley, and Elsburg Reefs lie in regular and invariable sequence above it. The Du Preez and other reefs lie in the Quartzite-Shale group, and therefore underlie the Main Reef, and the Black Reef occurs with the more recent dolomitic formation.

Trenching.—Where there is little depth of soil, a cross section may be obtained by cutting trenches across the formation deep enough to expose the underlying rock. Where the extent along the formation is considerable, such trenches are placed at regular distances apart.

It is better for one—generally the centre one—to be con-

tinuous throughout and to be kept ahead of the others, which are only opened up in places where, as it seems probable from the main line of section, reefs are likely to cross. Wherever a reef is struck, a small shaft is sunk so that the reef can be sampled where it is solid.

By these means any reef discovered is opened up in several places at regular distances apart, and a very correct idea of its grade can be obtained, for there is no doubt that, generally speaking, the banket reefs are as likely to improve along the surface as they are in depth. Where at all points a reef has shown a poor assay value from these small shafts, it is a reasonable assumption that it is equally poor in depth.

It is held by interested persons that in the Heidelberg district the reefs improve only in depth, but tabulated results do not confirm this opinion.

Where a reef has shown an irregular value, the shaft from which the best value was obtained is sunk deeper to see whether the high value is maintained.

Where the extent of the ground is small, one line of trenches may be sufficient, and should any reef be found, a shaft, sunk to a somewhat greater depth, say 100 feet, should, by careful sampling, prove sufficient to give a very good idea of the value of the reef. It is characteristic of the auriferous banket that a value so obtained may be considered to hold good for a considerable area of ground.

Diamond Drilling.—This method of prospecting is very valuable, especially for determining the location of a reef, and by it the continuity of the Main Reef to great depths has been proved.

In the East Rand district, where from the Ginsberg to the Blue Sky the banket formation is covered up by about 50 feet of sub-soil, the detritus of the coal measures, prospecting, principally by diamond drilling, has resulted in opening up a most valuable length of the extension of the Main Reef Series. In the Klerksdorp district, where igneous overflows

are abundant and extensive, the existence of several reefs has been determined or verified by the same means, and in prospecting the Black Reef the diamond drill has been used extensively in the endeavour to define the extent of the shoots of better-grade ore. Although in many places where bore-holes have been put down, the Black Reef has been judged not to be deeper than could be reached easily by a prospecting shaft, yet the amount of water which is often met with in the dolomite above this reef prevents the ordinary rate of shaft-sinking from being attained, and turns the balance in favour of prospecting by diamond drills.

It is, however, in proving the continuity of the reefs in depth that diamond drilling has been of the greatest use along the Rand, and in these deep bore-holes, sections of the formation are obtained which give the best possible idea of the exact distances between the various reefs, for the majority of faults and dykes dip at a high angle, so that in any vertical depth a less amount of disturbance is encountered than in an equal length along a horizontal plane.

Construction of Diamond Drill.—The diamond drill consists essentially of a steel tube, rotated by hand or machine power, the lower edge of which is armed with a crown set with diamonds; the latter form teeth of excessive hardness, which grind a circular groove into whatever rock may be met with, leaving a central column or core of rock projecting into the tube. This core is broken off from time to time and gripped by suitable appliances, which enable it to be brought to the surface, and thus to indicate the exact nature of the ground passed through. A stream of water is forced down through the tube and rises up outside it in order to keep the face of rock in the groove clear of fragments. It is also necessary to provide an arrangement for pressing the boring tube down when a hole is started, and for taking off a part of its weight when great depth has been attained. The various methods in which this is performed and in which the drill is

driven, constitute the chief differences between the various makes of diamond drill. Those most in favour on the Rand are the Sullivan, the Bullock and the American Rock Drill Company's drill, and for hand-drilling, either the hand diamond drills of the above makers or Alfred Short's hand-power diamond drill, manufactured in Durban.

The Sullivan drill has been used for some of the most important bore-holes on the Rand. In these drills the power is supplied by means of either one or two vertical cylinders actuated by steam or compressed air. The power is transmitted by a horizontal shaft and bevel gearing to the hollow drill rods. The feed in all the larger drills is hydraulic. A single hydraulic cylinder is employed, through the hollow piston rod of which the drill rods pass, the upwards or downwards pressure of the hydraulic piston being communicated by ball bearings, which are practically frictionless. Fig. 40 shows one of these drills, known as size "B," which was used in drilling, amongst others, the Henry Nourse Deep level bore-hole to a depth of some 800 feet. The machine is capable of drilling to a depth of 3000 feet, the diameter of the hole being 2 inches, and of the core $1\frac{3}{8}$ inches. The largest drill as yet used on the Rand is a Sullivan "P" drill, capable of drilling to a depth of 4000 feet, and bringing up a $1\frac{7}{8}$-inch core; this size was used for the Bezuidenville bore-hole. The Bullock "Dauntless" drill was used for putting down the Rand Victoria bore-hole to a depth of 2400 feet; it makes a 2-inch hole and brings up a $1\frac{7}{16}$-inch core. The engine here consists of two cylinders placed diagonally, the feed being a differential screw feed, capable of being regulated to three different speeds of boring.

The American Rock Drill Company's drills No. 3 and 7 are both used in South Africa; the former, shown in Fig. 41, is worked with fixed inclined cylinders, and has a hydraulic feed, consisting of a pair of hydraulic cylinders, the piston rods of which are attached to a cross-head, which in turn

transmits the pressure to the drill rods. This machine is intended for boring to a depth of 1500 feet, making a 2-inch hole and producing a $1\frac{3}{8}$-inch core. The No. 7 drill produces

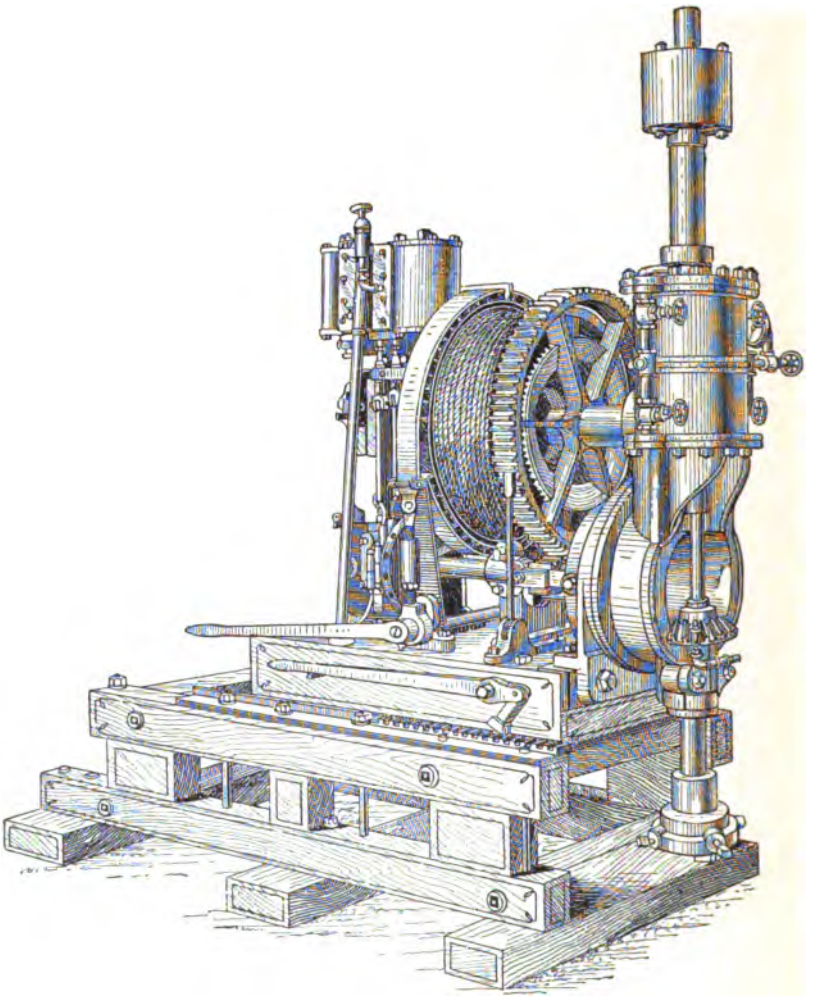


FIG. 40.—Sullivan drill, size "B."

holes and cores of the same diameter, but is intended only to drill to depths of 1000 feet. This drill is worked by oscillating cylinders, the mode of feeding being identical with the last named.

The M. C. Bullock Manufacturing Company of Chicago

makes various types of drills, the best known of which is perhaps the "Dauntless," shown in Fig. 42. This is the drill

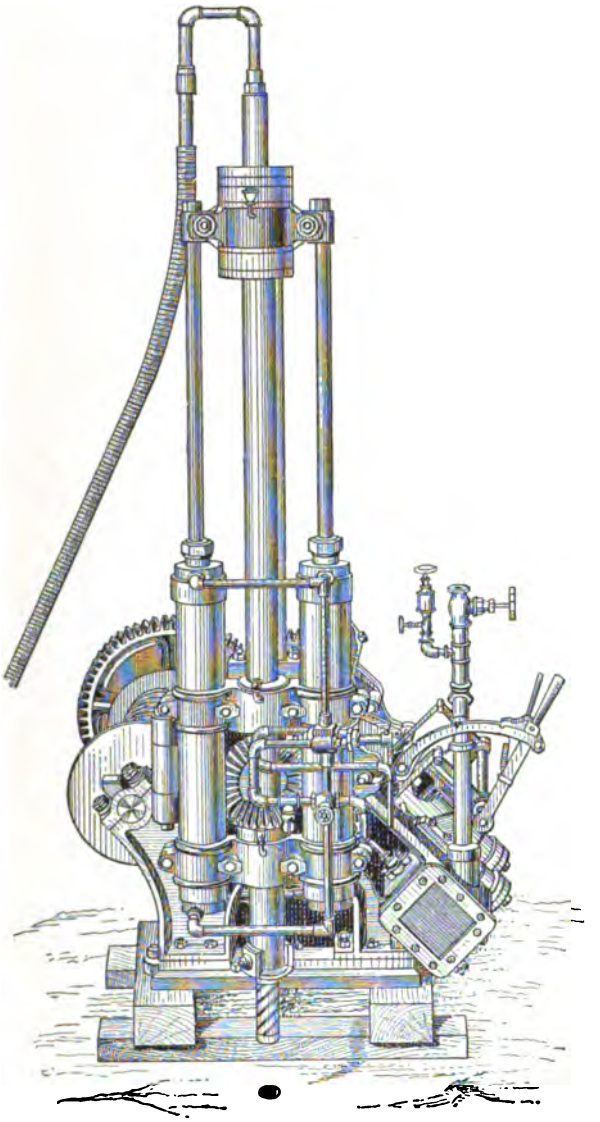


FIG. 41.—The American Rock Drill Company's No. 3 drill.

with which the famous Rand Victoria bore-hole was put down. It is actuated by a pair of 5 inch × 5 inch diagonal cylinders, which can also be arranged to drive the hoisting barrel. The

drill has a positive differential feed motion acting upon the

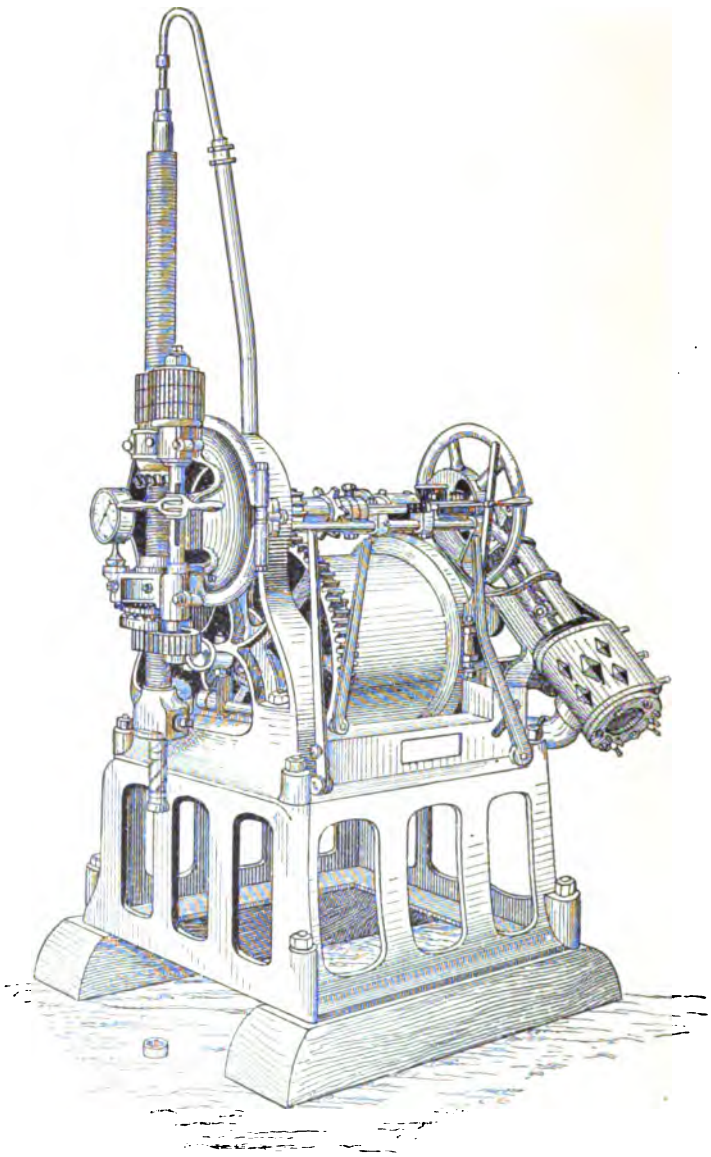


FIG. 42.—The M. C. Bullock drill "Dauntless."

long screw thread shown in the figure, and is mounted upon a swivel head. By means of hydraulic plungers the pressure upon the face of the drill is communicated to a pressure gauge,

which thus forms a thrust indicator, characteristic of all the

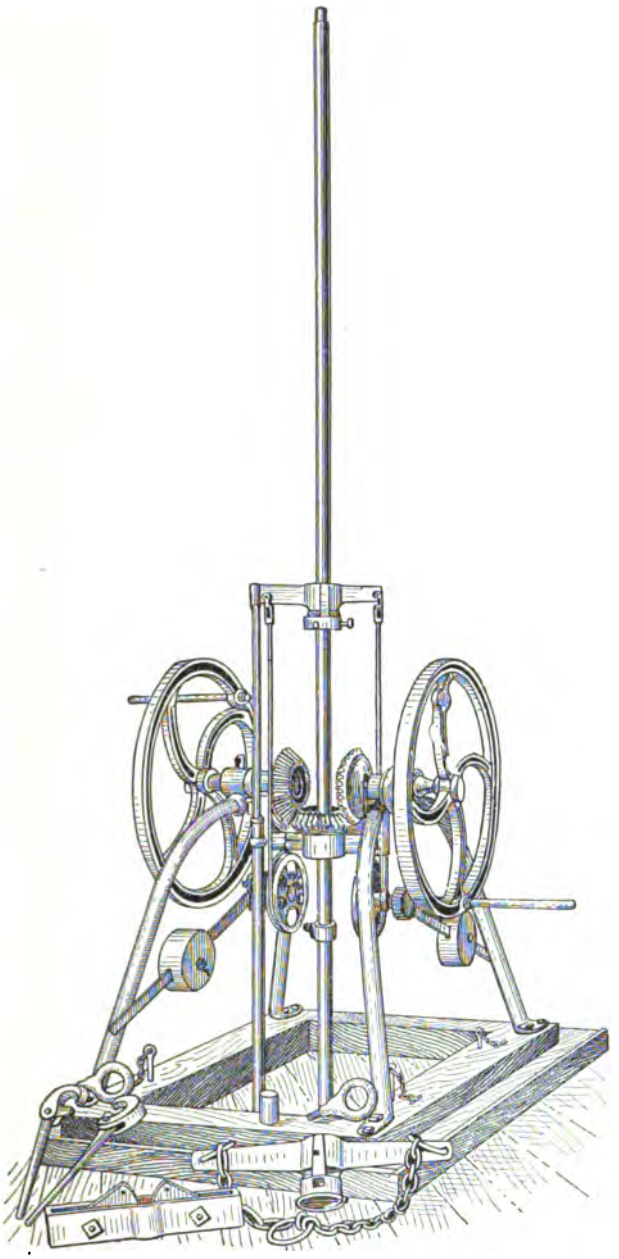


FIG. 43.—Short's hand-power drill.

Bullock drills. Most of these have the screw feed as shown,

but some use a simple two-cylinder hydraulic feed. The drill is worked by a 12 H.P. nominal boiler, and its guaranteed capacity is as follows :—

$2\frac{7}{16}$-inch hole.	1-inch core.	depth 2000 feet.
$2\frac{5}{8}$,,	2 ,,	,, 1500 ,,
$3\frac{9}{16}$,,	$2\frac{7}{8}$,,	,, 1200 ,,

In the hand-power drills, depths of from 200 to 400 feet can be reached, with cores 1 inch to $1\frac{1}{4}$ inch in diameter. The drill is worked by men working fly-wheels, to which handles are attached: the feed is produced either by means of weighted levers or by differential screw motion. Short's hand-power drill, a popular drill on the Rand, is shown in Fig. 43.

Details of Bore-holes.—The following are the particulars of some of the more important deep-level bore-holes which have been put down :—

Lancaster.—West bore-hole, situated 1370 feet south of the Botha's outcrop :—

	Depth.	Length of Core of Banket.	Assay Value.
South Reef . .	748 ft. 4 in.	3 in.	126 dwts. 15 grs. per ton.
Reef . .	778 ,, 10 ,,	30 ,,	2 ,, 10 ,, ,,
Leader . .	783 ,,	7 ,,	4 ,, 12 ,, ,,
Botha's Reef .	784 ,, 6 ,,	30 ,,	27 ,, 18 ,, ,,
Leader . .	787 ,, 7 ,,	5 ,,	10 ,, 13 ,, ,,

East bore-hole, situated 1175 feet south of the Botha's outcrop :—

	Depth.	Length of Core.	Assay Value.
Reef . .	551 ft. 2 in.	11 in.	4 dwts. per ton.
Leader . .	553 ,, 6 ,,	3 ,,	Nil
Botha's Reef .	555 ,,	10 ,,	271 ,, 5 grs. per ton.

Crown Deep (Rand Deep Level), situated 1200 feet south of the South Reef outcrop :—

	Depth.	Length of Core.	Assay Value.
South Reef .	828 ft.	18 in.	11 dwts. 12 grs. per ton.
Main Reef Leader	895 ,,	18 ,,	275 ,, ,,
Main Reef .	904 ,, 5 in.	54 ,,	10 ,, ,,
North Reef .	957 ,,	24 ,,	2 ,, ,,

Bezuidenville, 5080 feet from the outcrop, with a total depth of 3728 feet, struck the Bird Reef Series between the depths of 1100 and 1400 feet, and the Main Reef Series between 3150 and 3250 feet. This is the deepest bore-hole as yet put down on the Rand.

Nourse Deep, situated about 300 feet from the outcrop of the South Reef:—

	Depth.	Length of Core.	Assay Value.
South Reef . .	605 ft.	30 in.	112 dwts. 4 grs. per ton.
Main Reef Leader .	659 „	30 „	21 „ „

Rand Victoria, situated 4100 feet from the outcrop:—

	Depth.	Length of Core.	Assay Value.
Reef . . .	2343 ft.	13 in.	23 dwts. 20 grs. per ton.
Reef . . .	2391 „	9 „	23 „ 20 „ „

Rose Deep (Umbilo block), situated 1080 feet from the outcrop:—

	Depth.	Length of Core.	Assay Value.
Reef . . .	754 ft.	20 in.	38 dwts. 1 gr. per ton.
Reef . . .	760 „	11 „	10 „ 18 „ „
Reef . . .	777 „	21 „	49 „ 9 „ „

Simmer East, Lohse shaft, situated 5080 feet from the outcrop:—

	Depth.	Length of Core.	Assay Value.
Reef	1849 ft.	19 in.	20 dwts. per ton.
Reef	1851 „	4 „	40 „ „
Reef	1865 „	9 „	40 „ „

This bore-hole was started from a depth of 693 feet in the shaft.

Chimes Mines, situated 4340 feet from the outcrop:—

	Depth.	Length of Core.	Assay Value.
Main Reef Leader .	1738 ft.	4 in.	14 dwts. per ton.
Main Reef . .	1754 „	36 „	16 „

Assay Value of Reef Cores.—As a means of comparing the results from bore-holes with those obtained in development, the following instances are instructive:—

In the north cross-cut, from No. 1 shaft of the Lancaster, the lower portion of the Botha's Reef averaged from both sides of the cross-cut 10 inches thick and 216 dwts. 18 grs. per ton assay value; this point was 110 feet east of and 10 feet below the point where the same reef was cut in the eastern bore-hole, its thickness there being 10 inches and assay value 275 dwts. 5 grs.

No. 1 shaft of the Crown Deep is 200 feet away from a bore-hole mentioned above, and in it the South Reef was struck, at a depth of about 935 feet, 48 inches thick, of which the upper 42 inches were poor and the lower 6 inches assayed 28½ dwts. per ton. In the bore-hole it was 42 inches thick, of which the upper 32 inches were poor and the lower 10 inches assayed 24 dwts., and out of the total width 18 inches assayed 11 dwts. 12 grs. per ton, as stated above.

The Main Reef Leader was struck in this shaft 12 inches thick and 179½ dwts. per ton in assay value, and in the bore-hole 18 inches thick and 275 dwts. to the ton, assay value. These two cases show the bore-hole results to have afforded a correct idea of the grade of the ore.

In submitting the cores to assay it is generally required that only half be ground up, the remainder being kept intact for reference. To effect this the core to be sampled is placed on a smooth, flat iron surface and marked longitudinally with a cold chisel. A strong line having thus been made, the core is struck sharp hard blows with the chisel along the line, when, if it be of banket or quartzite, it breaks very cleanly in halves, though shale or slate cores do not break so well. In the case of one reef, of which the core was obtained in three parts, each set of halves so obtained was assayed separately, with the following results :—

	First Set.	Duplicate Set.
1 . . .	12 dwts. 19 grs.	13 dwts. 10 grs.
2 . . .	1 „ ...	... 12 „
3 . . .	11 „ 5 „	11 „ 2 „

This result illustrates the accuracy of this method of dealing with cores.

The length of the core through a reef being given, the actual thickness of the reef can be calculated from the known dip.

Depth and Dip from Reef Cores.—The depth and dip of a reef which are indicated by a bore-hole may be incorrect, owing to the liability of the bore-hole to deviate from the perpendicular.

Deviation has been experienced in several cases in other countries, but has not yet made itself felt on the Rand, though in one or two cases, for example in the Ferreira, the bottoms of bore-holes have been encountered in subsequent mining.

It is stated, as showing the flexibility of the rods, that a length of 2000 feet of small rods screwed up can be bent into a complete circle.

In all sedimentary strata there are layers of different hardness. This being so, where they are inclined, as along the Rand, one segment of the boring bit may be drilling in rock of different hardness from the other segment. If in softer rock, the tendency will be for the bit to incline to that side, and *vice versa*, so that with inclined sedimentary beds there would appear to be a greater tendency to deviate across than along the formation, and the amount would be to some extent proportionate to the dip, for when the strata are flat the chances of deviation are at a minimum. Deviation would thus appear to occur on much the same lines as are followed in the refraction of a ray of light passing obliquely through media of different densities. A deviation when once formed would appear to predispose to another, though when passing beds where the relative hardnesses are in an order inverse to that which caused the deviation, there should be a tendency to rectification. The depth at which a reef is indicated to be by a bore-hole is not so likely to be materially different from the true depth as the dip may be from the true dip, and as the

dip of the beds bears greatly on the position of the reef in the deep levels, this is a most important consideration.

Should the deviation be along the strike, the dip given by the bedding planes is not materially disturbed; but should it be across the formation, which, as stated before, would likely occur more often, it is liable to be a good deal out; thus, if the line of the bore-hole for a length near the bottom be 15° off the perpendicular, and in a direction opposite to the dip of the beds, the angle as given by the cores would be 15° too flat, and *vice versa*, so that if the actual dip of the beds is 15° and the deviation also is 15°, the cores would register an angle of 0° in the first instance and one of 30° in the opposite case, as shown in the following diagram (Fig. 44).

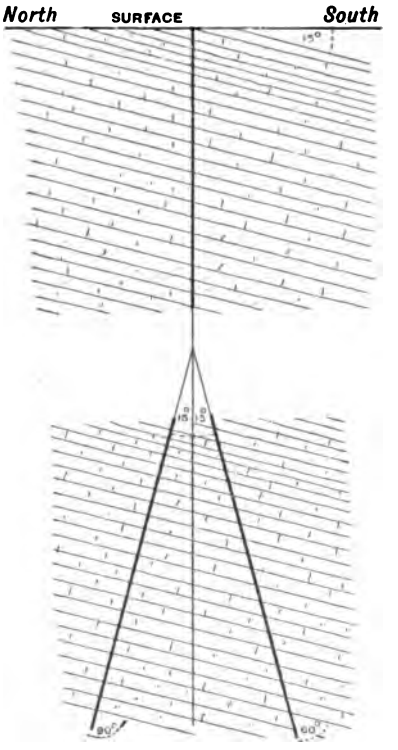


FIG. 44.—Diagram illustrating the effect of deviation from the perpendicular upon the angle which the bedding planes make with the direction of the bore-hole.

Earth Temperatures from Bore-holes. — Mr. Hamilton Smith made some determinations on the temperature at different depths in the Rand Victoria bore-hole, which, however, he afterwards stated, were incorrect, indicating too great a rise in depth owing to inaccuracies in the thermometers used. Bore-holes are always full of water, and it is the temperature of this which is taken at different depths. The attainment of thoroughly accurate results requires that the water be quiescent. This is practically the state of the water in most of the bore-holes put down south of the outcrop along the Witwatersrand, though where extreme accuracy is required

special apparatus must be used to prevent the formation of any currents in the column of water. Owing to the pressure of the water, special thermometers have to be employed.

The Surveying of Bore-holes.—In a paper with this title, read before the South African Association of Engineers and Architects on 28th July 1897, Mr. Andrew F. Crosse detailed a method of surveying bore-holes and described his apparatus. From this paper the following notes are taken :—

The apparatus consists of a multiplying winch, on which is wound a sufficient length of highly tempered steel wire, No. 19 B.W.G., with a breaking strain of over 450 lbs. This wire passes over a drum having a circumference such that one coil of wire on it is exactly equal to one foot in length, and every revolution of this drum is indicated on a dial, so that the exact depth of the end of the wire can be accurately shown. To this end there is attached by a loop and swivel arrangement a brass tube 2 feet long, which can be sealed hermetically at either end. The survey consists in determining the altitude, that is the angle from the horizontal, and the azimuth, that is the angle from the magnetic north, of this brass tube at different depths.

Inside of it two other brass tubes fit accurately, one at a time, in a position determined by longitudinal grooves. Each has a small bar about the size of a needle fixed at its base, and each can be hermetically closed by caps at either end. On one of these bars a tiny circular mirror is accurately balanced, so that within certain limits it is always horizontal, and on the other a little magnet swings, having two aluminium pins pointing downwards, one from each end.

A short time after these have been lowered down to any point in the bore-hole, the mirror settles into a horizontal plane, the magnet points north, and the position of the tube is that of the bore-hole at that point. These positions are fixed by partly filling the inside tubes with a liquid mixture, which after a certain time—an hour at ordinary temperatures

—sufficient for the needle and mirror to come to rest, sets hard.

When drawn to surface it is easy to measure the angles which the tube makes with the magnet and the mirror, and these angles are the azimuth and altitude respectively of the tube when it was in position in the bore-hole.

Where the strata are highly inclined, a section across the formation is obtained quicker and at less cost by setting the drill at an angle to the formation. In such cases, however, there is a limit to the depth of hole which can be drilled, determined by the deviation from the angle at which the hole started, which constantly increases, owing to the weight of the rods.

Prospecting underground by diamond drilling is sometimes undertaken in preference to cross-cuts, because it is more speedy and less costly. In the north cross-cut of the Glen Deep, on the 923 foot level, about 40 feet west of the shaft, a hole was drilled at an angle down to strike the reefs, in which it was successful. In the Rose Deep and others of the deep-level mines the hand diamond drill is in frequent use underground.

In No. 2 shaft of the Rietfontein A, when the reef was lost on the 3rd level the shaft was continued, and on the 4th level a bore-hole was put down without, however, finding the reef.

The Lohse shaft bore-hole of the Simmer East was started at the bottom of the shaft, which was then 693 feet deep. As this shaft is now being continued, some interesting information should be obtained about the course of that bore-hole.

Records of Bore-hole Results.—When a bore-hole is being put down, a man who is regarded as a confidential and responsible agent of his employer is engaged to receive and arrange the core, and to keep a record of all the work done. Each piece of core as it comes up is best marked by consecutive numbering from the top downwards. This may be done

on a gummed label, but it is better to boldly mark the figures on the core itself with red or other bright-coloured enamel, because such pieces, when dry, can then be dipped into water for the purpose of examination. There are several kinds of boxes used for receiving the core; some are long and narrow, holding only two rows; some have a number of sliding trays, and others of lifting trays. Of these, the latter kind are the best, because they can be made to hold a convenient amount and the trays are easily lifted, whereas with the sliding trays the core when disarranged, as it is always apt to be, catches the trays, so that it is difficult to draw them out. The best length for a box is about 5 feet inside, and it should be wide enough to hold from three to five rows depending upon the diameter of the core. The bottom of the box is arranged to form one tray, and as another fits inside with a convenient amount of clearance, it is made to take one row less. Such boxes hold from 35 to 40 feet of core, which makes a convenient weight for handling. The several rows are kept separate by lengths of beading set at regular distances apart.

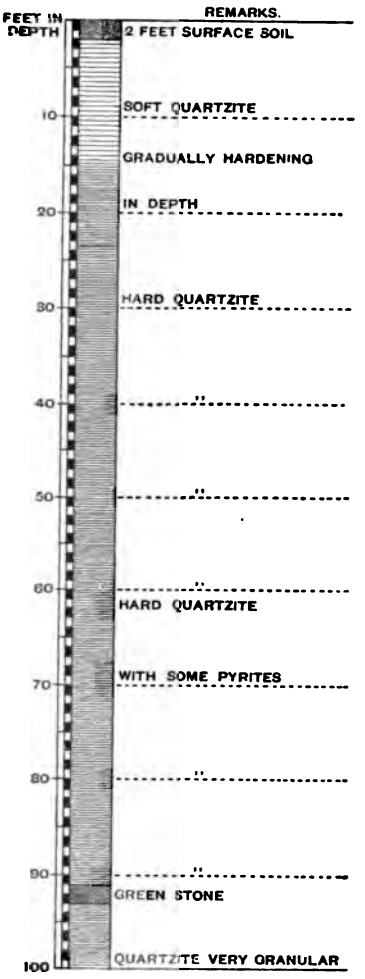


FIG. 45.—Graphic record of a bore-hole.

In the case of the Bezuidenville bore-hole the core was arranged on fixed shelves to facilitate examination.

In addition to preserving the core in boxes, a record is

kept on sheets, a specimen of which is shown on p. 149, and a graphic representation to scale, such as is shown in Fig. 45, is also sometimes made.

The following is a comprehensive statement of bore-hole results and costs taken from an actual bore-hole :—

BORING RETURNS—BORE-HOLE NO. 4

Gross Time occupied

10th February to 30th March 1896, 49 days = 1176 hours, of which :—

Time working	29 days
Time not working	$19\frac{1}{3}$,,
Time erecting	$\frac{2}{3}$,,
	49 ,,

Time working, 29 Days

Time boring (routine work)	$27\frac{1}{2}$ days
Repairs, etc.	$1\frac{1}{2}$,,
	29 ,,
Total depth	$596\frac{1}{2}$ feet
Average rate per working day	20.57 ,,

Time not working

Sundays	6 days	= 144 hours
Want of water	2 ,,	= 24 ,,
Want of coal	2 ,,	= 24 ,,
No night shift and off work	$9\frac{1}{3}$,,	= 224 ,,
	$19\frac{1}{3}$,,	= 464 ,,

Boring through Surface Soil

Total working time	3 days	= 72 hours
Accident repairs	$\frac{1}{2}$,,	= 12 ,,
Total boring time	$2\frac{1}{2}$,,	= 60 ,,
Depth sunk		79 feet
Average rate per working day		26.4 ,,
Amount of tubing		$79\frac{1}{2}$,,
Time occupied fixing tubing		12 hours

Boring through Solid Formation

Total working time	26 days
Accident repairs	1 „
Total boring time	25 „
Depth	517½ feet
Average rate per working day	19.9 „
Number of lifts	60
Total depth	596½ feet
Average depth bored each lift	9.8 „
Core recovered	510 „
Percentage of core recovered	83.5 per cent
„ in shale	98.75 „
„ in quartzite	97.30 „
„ in igneous rock	88.70 „
„ in coal measures	71.00 „
Average depth drilled per working day	20.57 feet
„ in surface formation	26.33 „
„ in solid „	19.90 „
Amount of tubing	79.50 „
Diameter of bore-hole	2 1/16 inches
„ core	1⅜ „
Name of drill	Bullock "Champion"
Power „	8 H.P.
Average daily water-supply	1440 gallons
Number of white men employed	3
„ black „ „	4

Costs.

Contractor, 500 feet, at 30s.	£750 0 0	
„ 97 „ at 35s.	169 15 0	
		£919 15 0
Water-supply—		
Native wages	£24 2 8	
Food and expenses	4 8 9	
White wages	40 12 6	
Stores	5 14 7	
		74 18 6
Core-watcher		40 12 6
Sundries, core-boxes, etc.		29 5 0
Hire of drill		50 0 0
		£1114 11 0

Total depth	597 feet
Cost per foot	£1 17 4

The average cost of eight bore-holes put down by the author, with a total depth of 2686 feet, was 36s. 6d. per foot, which includes the full costs under the heads indicated in the above statement.

The following are some figures relative to the Bezuidenville bore-hole :—

Total depth	. . .	3728 feet
Average rate per working day	.	21.1 „
Percentage of core recovered	.	91.75 per cent

The cost, including money paid to contractors, water-supply, supervision, and depreciation of plant and buildings, was 50s. 6d. per foot.

The best method of keeping a systematic record of boring operations is to enter all information as obtained in a core book; this is best divided into heads, as shown by the subjoined specimen sheet.

Prospecting by Cross-cuts.—When the reef is lost underground it is generally picked up again by cross-cuts rather than by winzes or shafts. The long cross-cuts in the George Goch and Langlaagte United, respectively 740 feet and 687 feet in length, the latter being shown on p. 117, illustrate the value of cross-cuts in this connection. The only case where the faulted portion of the reef cannot be picked up by a cross-cut on the same level is where the fault runs parallel with the formation and throws the reef down normally. Notable instances of this have occurred in the Village Main Reef, where the reef was picked up by sinking winzes on the other side of the fault plane, and in the Rietfontein A, where the shaft was continued deeper and cross-cuts put in at intervals of 100 feet, till the reef was struck.

Prospecting Shafts.—The shafts that are sunk on reefs which have been discovered in prospecting, to determine their value, are kept small. The reefs in the eastern portion of the Blue Sky were discovered by sinking a vertical shaft down to the solid formation and cross-cutting.

Specimen Sheet for Core Book

Date.	Number of Lift.	Depth at Bottom.		Depth Drilled since Last Lift.		Length of Core obtained.		Number of Core Pieces.	Length of Core Pieces.		Remarks.
		Feet.	Inches.	Feet.	Inches.	Feet.	Inches.		Feet.	Inches.	Dip, nature of rock, assay value, etc.

CHAPTER VIII

SHAFTS, SINKING AND TIMBERING

Location of Shafts.—The shafts of the outcrop mines are chiefly incline shafts, which are sunk to follow as nearly as possible the inclination of the reef. In a good many cases they are started on the reef and continued as nearly as possible on it, but there would appear to be an advantage in keeping the shaft some little distance in the footwall of the reef, because then it would be undisturbed by the workings, and all the ore, which otherwise would have to be left as shaft pillars, could be taken out. Owing to dislocation by faults, it is not possible to keep the shaft on the reef continuously for any length, or to maintain a constant distance from it. In view of this, it is now more usual with Main Incline shafts to set them at such a constant angle, as appears from all the information obtainable, to be the most likely to keep the shaft pretty close to the reef throughout its extent.

The Main Incline shaft of the City and Suburban has been sunk to its present depth of about 2000 feet at an angle of 30°. It was started at the surface some distance in the footwall of the reef, which was dipping more steeply, but in the lower levels it is now some distance in the hanging wall, as shown in Fig. 80. It is likely that deeper still the reefs will approach the shaft again.

The two Main Inclines of the Robinson Mine were kept from surface down through the upper levels at a constant

inclination, which in the lower levels was changed to a flatter angle, because the reefs had gradually got away into the hanging wall.

The Main Incline shaft of the Durban-Roodepoort has, down to its present depth, changed its inclination four or five times through small angles.

There are very few of the outcrop mines which are worked by vertical shafts. The costs given in the chapter on hoisting show that the system of sinking vertical shafts to the reef and then incline shafts on the reef is more costly for hoisting than continuous hoisting through an incline shaft. The Langlaagte Estate was opened up by three shallow vertical shafts down to the reef, through which the rock was hoisted in cages. As the deeper levels of this mine became opened up it was found necessary to sink another and larger vertical shaft at a point farther to the dip.

There are many outcrop mines which, starting with shallow vertical shafts, afterwards adopted incline shafts. This is well instanced in the Robinson mine, which was opened up first by three shallow vertical shafts, exactly similar to those of the Langlaagte Estate; these were afterwards abandoned and two Main Incline shafts sunk instead.

In consequence of the comparatively small holdings of the outcrop companies the shafts are placed unnecessarily close together. Where, however, several small blocks have been consolidated and a new system of development adopted, the shafts are placed at distances apart which are comparable to those determined upon in the deep-level mines. Thus the three incline shafts of the Bantjes Consolidated are situated about 2400 feet apart, and those of the Main Reef Consolidated are at almost equally large distances apart.

In the first row of deep levels the shafts are situated at an average distance of about 1700 feet from the outcrop of the reef, at which distance the average vertical depth of the reef is about 1100 feet. These shafts are placed at an average dis-

tance of 370 feet from the northern boundary of the property, and at about 2280 feet from the southern boundary. There are generally two shafts on each of these properties, and of the two one is often placed farther to the dip, so that, being the deeper, it may be used for a main pumping shaft, and also because natural ventilation is thus promoted. The average area of one of these deep-level properties on the first row being about 220 claims, each shaft is laid out to work about 100 claims, which is about twice as much as the outcrop shafts are intended for. They are situated on any one property about 1600 feet from one another along the strike, and they are placed symmetrically with regard to the eastern and western boundaries. The shafts of the second row of deep levels are situated on an average about 5000 feet from the outcrop of the reef, and about 1000 feet from the northern boundary of the properties. At this distance from the outcrop the average depth of the reef, if undisturbed, should be about 3000 feet, but owing to the influence of reverse faults, accompanied by upthrows on the south side, this depth will probably be considerably less. The average claim area of these properties is close upon 500 claims, so that each shaft is laid out to work about 200 claims. The object of placing them so far from the northern boundaries of the properties is that there shall not be too great a length of incline below the vertical shaft; otherwise, with the large extent of ground to the dip which these properties own, it might become necessary to put down another row of vertical shafts.

Size of Vertical Shafts.—The size of a shaft depends primarily upon the amount of ore which it is estimated will be drawn through it. In the cases of those mines (the deep levels) where, for a considerable time at least, connection with the surface will be effected only through the shafts, adequate space must be allowed in the pump and ladder-way for the transmission of all the power, electricity, compressed air or steam, which is to be employed underground, and there must

be an open area sufficient for proper ventilation. In these deep mines the number of shafts is limited by considerations of cost, and, in consequence, on the deeper reefs the number of shafts will be less, and the size of each shaft will be greater.

The following are the ruling sizes of vertical shafts along the Witwatersrand :—

(1) Those which are on ground in which the reef outcrops. These are rarely more than 350 feet deep to the reef, and there are very few of them.

Length of shaft inside timbers	12 feet
Width " "	5 "
Number of hoisting compartments . . .	2
Length " " . . .	4 feet
Length of pump and ladder-way . . .	3 "

Where there is only one shaft, the compartments are larger, as is the case with the Spes Bona, which has three compartments, and is 16 feet long and 5 feet wide.

(2) Those which are on the first row of deep-level ground. These shafts strike the reef after it has passed out of the outcrop ground at depths generally from 600 feet to 1600 feet.

Length of shaft inside timbers . .	16 feet or 21 feet
Width " " . .	6 "
Number of hoisting compartments .	3 or 4
Length " " .	4 feet or 4½ feet
Length of pump and ladder-way .	6 "

No. 2 shaft of the New Modderfontein, which is included in this group, has five compartments, four of which are for hoisting, with a total length inside timbers of 24 feet and a width of 6 feet. This shaft was made this size because two reefs were to be intersected at such a distance apart that practically two mines would be worked from the shaft. The upper reef, the Chimes, was struck at 560 feet, and at about 610 feet two of the hoisting compartments were stopped, whilst the other two, together with the pump and ladder-way,

were continued till the Van Ryn Reef was passed through at from 1110 feet to 1113 feet.

For similar reasons the two shafts of the Witwatersrand Deep were made with five compartments, and in these shafts the pump and ladder-way is somewhat larger.

(3) Those shafts which are on ground underneath which the reef lies at greater depths, 2000 feet and over.

Length inside timbers . . .	26 feet
Width " " . . .	6 "
Number of hoisting compartments .	4
Length " " . .	4½ feet
Length of pump and ladder-way .	6½ "

The following are some actual sizes of shafts :—

(1) Langlaagte Estate, Vernon Shaft; Outcrop Company.
(2) Durban-Roodepoort Deep, No. 2 Shaft; Deep Level Company.
(3) Jumpers Deep, No 2 Shaft; Deep Level Company.
(4) Simmer West, Howard Shaft; Deeper Level Company.

	(1)	(2)	(3)	(4)
Length of shaft inside timbers .	13 ft.	16 ft.	21⅓ ft.	25 ft.
Width " " " .	5½ "	6 "	6 "	6 "
No. of hoisting compartments .	2	2	3	4
Length of hoisting " .	4 ft.	4½ ft.	4⅔ ft.	4 ft.
Length of pump and ladder-way .	4 "	6 "	6 "	7 "
Depth of reef	340 "	1314 "	1245 "	3000[1] "

The Bonanza shaft is unusually wide, this dimension being 6½ feet, and the hoisting compartments of the Roodepoort Central Deep are each 5 feet long, which is also unusual. The size of the hoisting compartments may be said to be much the same throughout; that of the pump and ladder-way increases in the deeper shafts, but when one shaft is larger than another, this is mostly due to its having more hoisting compartments.

[1] Estimated.

The sizes given are those inside the main timbers or "in the clear," as it is termed; the actual size of the excavation is larger by about 2 feet each way.

Size of Incline Shafts.—The average size of incline shafts is about 16 feet in length and $5\frac{1}{2}$ feet in width inside the main timbers, and the length is usually divided into two hoisting compartments of $4\frac{1}{2}$ feet each and one pump and ladder-way of 6 feet.

The three incline shafts of the Consolidated Main Reef are of four compartments, with a total length of 20 feet and a width of 6 feet, and they have been laid out to work the property, which has an extent of 7000 feet to the dip. The shafts of the Bantjes Consolidated, which have also been laid out to work a similar extent of ground, are only of average size.

The City and Suburban Main Incline, which is now down over 2000 feet, is also 20 feet long and $5\frac{1}{2}$ feet in width.

Sinking.—This term is taken to mean the removal of the ground so as to advance the bottom of a shaft.

Along the Witwatersrand, sinking, except when through soft and watery decomposed igneous rock near the surface, is always done by blasting by means of charges of explosives contained in drilled holes.

The deep-level shafts are placed with their long sides parallel to the formation, the single exception being the main shaft of the Village Main Reef, which is placed across it. The most central compartment is generally used as the "sinking compartment," through which all the hoisting is done in sinking, and below which the shaft is first advanced.

There are three principal methods of breaking the bottom of a shaft, viz.:—

(1) By drilling over the whole face in one shift, blasting the centre holes for the "cut," and then squaring up.

(2) Drilling the ends after the cut has been blasted.

(3) By placing the holes to suit the formation.

(1) This method is only used with machine drills.

The following is a description of the procedure at the Vogelstruis Consolidated Deep, from information supplied me by the manager, Mr. A. R. Robertson.

The excavation is about 18½ feet long and 7½ feet wide,

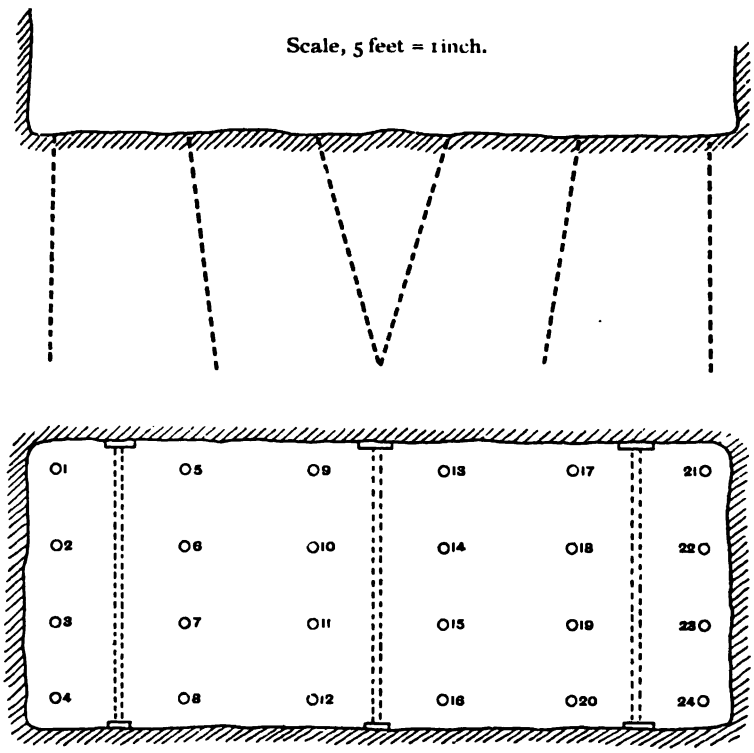


FIG. 46.—Diagram showing the arrangement of the holes in the bottom of the Vogelstruis Deep Shafts working with machine drills.

the size of the shaft between timbers being 16 feet by 5½ feet, length and width respectively.

The bottom is drilled over with twenty-four holes placed as shown in Fig. 46.

These are drilled by machine drills clamped on to stretcher bars, the positions of which are indicated in the diagram by dotted lines; each machine drills eight holes, four on either side of its bar. The holes of the two centre rows, which

constitute the "cut" proper, are inclined so as to meet in depth, each with the one opposite to it; those also of the two intermediate rows are inclined towards the centre, whilst those of the two outside rows are as nearly vertical as possible. The inside holes Nos. 5 to 20 are about 6½ feet deep; they are blasted together and before the outside holes, so that the centre of the shaft is brought away to a depth of about 4½ feet. After a certain amount of cleaning up has been done, the outside holes Nos. 1 to 4 and 21 to 24 are blasted, and the bottom is once more squared up.

This is generally done once every twenty-seven hours in three shifts. The first is occupied in drilling, which generally takes over six hours, after which the inside holes are blasted. After an interval of about ten minutes the second shift proceeds down to clean up, and at the end of this shift the outside holes are blasted; this generally takes another ten hours. After a second interval of ten minutes the third shift goes down and cleans up the bottom, so that drilling may start again.

During December 1896 the centre shaft of the property was sunk 141 feet and timbered 147 feet. Four drillmen and four white helpers were engaged during each drilling shift, the former receiving wages at the rate of thirty pounds a month and a bonus of ten shillings per foot for every foot sunk above 70 feet, and the latter receiving fifteen shillings per shift, with a bonus of half-a-crown for every such foot.

The cleaners were paid fifteen shillings per shift, and whenever they completed their work under ten hours they received half-a-crown extra.

Altogether 2906 feet had been sunk on this property up to the 31st May 1897, the costs of sinking which are given in the tabulated statement on p. 191.

The following is a description of the procedure at the Durban-Roodepoort Deep, from information supplied me by the manager, Mr. F. H. P. Cresswell.

The shafts are 16 feet long by 6 feet wide in the clear, and the size of the excavation is about 18½ feet by 8 feet.

The bottom is drilled over with twenty-one holes, placed in seven rows of three each, four rows being on one side of the sink and three rows on the other. This arrangement is made to suit the position of the sinking compartment, which

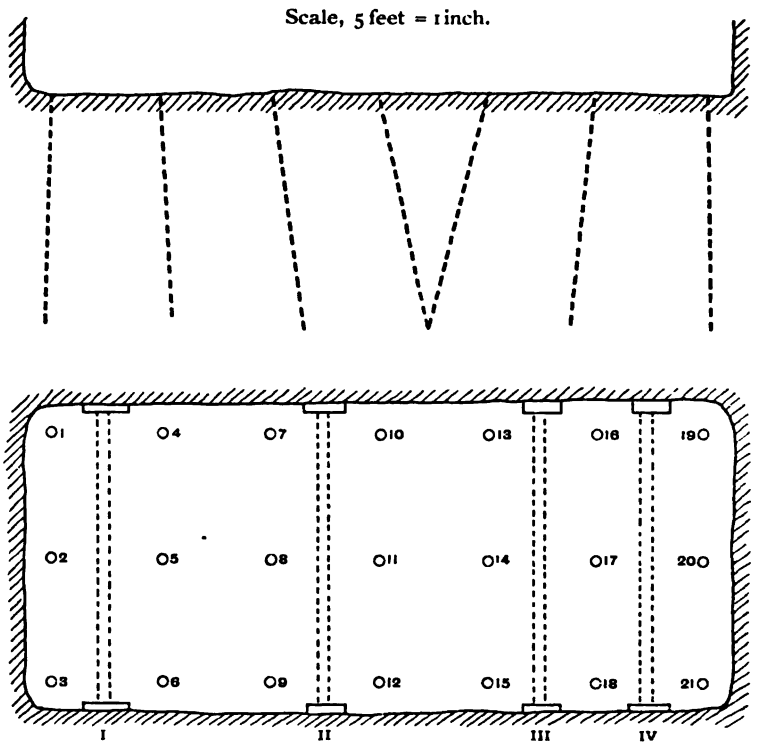


FIG. 47.—Diagram showing the arrangement of the holes in the bottom of the Durban-Roodepoort Deep Shafts.

is not quite central. The position and inclination of the holes are shown in Fig. 47, where the dotted lines indicate the positions of the bars, of which four are in use; from No. I. bar the holes 1, 2, 3 are first drilled, and then the machine is swung over and 4, 5, 6 are drilled. From No. II. bar the holes 7, 8, 9 are first drilled, and then numbers 10, 11, 12. From No. III. bar the holes 13, 14, 15, and then 16, are drilled, and from No. IV. bar, numbers 17 and 18, and then

19, 20, 21. This apportionment of holes is made because those drills which are on the bars I. and II. can get to work the first, and that on No. III. bar the last.

The holes 10 to 15 represent the cut proper, and these are inclined so that if possible they meet, each with the hole opposite to it.

At the first blast fifteen holes, numbers 4 to 18, are fired; of these the cut holes, with shorter lengths of fuse, are arranged to go off first, and at the second blast the two end rows of holes are fired. Mr. Cresswell tried, for three rounds, to blast the whole number at once, arranging fuses of varying length so that the holes would go off in correct sequence. He found, however, that so much of the rock was broken into fines that it packed quite hard with the water under foot, rendering the process of cleaning up more difficult and lengthy.

Each squaring up of the bottom of the shaft, which is variously called a "round" or a "sink," occupied on an average about thirty-two hours, generally divided into four shifts. During July 1896, in No. 2 shaft twenty-seven rounds were made and 131 feet were sunk; the average time of each round was 27½ hours, made up as follows:—

		Hrs.	Min.
1st shift	Drilling, from first drill going down to last drill coming up . .	5	15
	Charging and firing first blast .		40
	Allowed for smoke to clear away .	1	25
2nd shift	Clearing up, from skip going down to second blast	5	20
	Charging and firing second blast .		45
	Allowed for smoke to clear away .	1	20
3rd and 4th shifts	Clearing up, from skip going down to drills going down again . .	12	45
		27	30

In this same No. 2 shaft the following consecutive monthly sinkings were made:—

1896.	
May	110 feet
June	106 „
July	131 „
August	121 „
September	135 „
October	123 „
Six months	726 „

At the end of October the total depth of the shaft was 1410 feet. The cost of this sinking is given in the tabulated statement on p. 191.

The following interesting statements refer to the shafts upon the same property :—

DETAILS OF MAY SINKING 1896

	No. 1 Shaft.	No. 2 Shaft.
Total number of rounds	19	23
Average time drilling each round	10 hrs. 38 min.	7 hrs. 39 min.
Longest „ „ „ „	12 „ 40 „	12 „ 10 „
Shortest „ „ „ „	7 „ 20 „	5 „ 30 „
Average „ cleaning up	26 „	24 „ 7 „
Longest „ drilling and cleaning up	43 „ 30 „	52 „ 35 „
Shortest „ „ „ „ „	24 „ 10 „	25 „
Average „ „ „ „ „	36 „ 38 „	32 „ 21 „
Average depth sunk per round	4 feet	4 feet 9 inches
Total depth sunk during the month	75 „	110 „

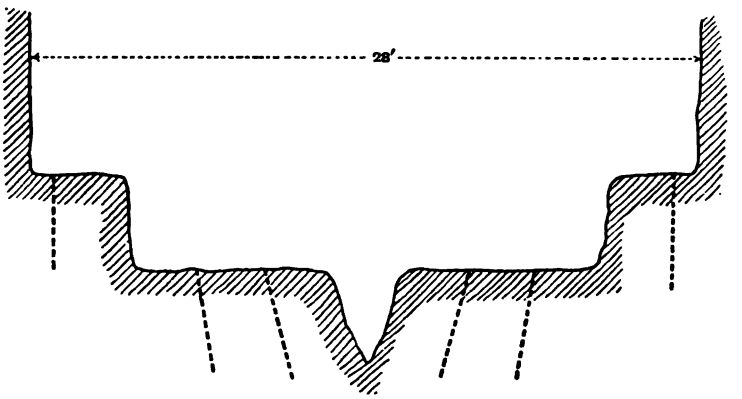
The average time for the whole round in the above statement is inclusive of everything except a stoppage of forty-eight hours for repairs to No. 1 winding engine. In order to get a correct idea of the time taken in cleaning up from this statement, two hours per round must be taken off as time for smoke to clear away, and about one and a half hours for delay caused by timbering.

(2) By this method the shaft is drilled over in sections; first the sink is drilled and blasted, and then the benches on either side.

The bottom of a shaft with the sink out and the benches drilled over is shown in Fig. 48.

As some amount of drilling is done on every shift, this method is not suitable for employment with machine drills, for a considerable amount of time would be lost in raising, lowering, and fixing them in position. It is, however, especially suitable for hand drilling, because it allows the day to be divided up into three shifts of eight hours each, and because all the holes are drilled downwards.

In the Clement shaft of the Simmer and Jack East,



Scale, 8 feet = 1 inch.

FIG. 48.—Diagrammatic section of the bottom of a large shaft which is being sunk by hand labour.

during July 1897, 156 feet were sunk by hand labour and 147 feet were timbered, bringing the total depth up to 1546 feet. One white man and, on an average, thirty-seven boys were down below each shift of eight hours; of these, one was a "boss" boy to look after the others and to assist in charging up, and two or three of the smaller boys were employed taking dulled drills to the shop and returning with sharp ones; on some shifts there were as many as forty-four boys. In drilling, the boys worked double-handed with 8 lb. hammers, one striking and the other holding the drill; the holes were put down about 5 feet or 5½ feet. The shaft was large, the excavation being about 28 feet long by 8 feet wide, and it

required about thirty holes and one and a half cases of gelatine to advance the bottom a complete sink.

To Mr. Sydney A. Chambers I am indebted for the following description of the record sinking at the Angelo Deep during the month of August 1897 :—

"Sunk West Shaft during August 164 feet; timbered 150 feet. The depth of shaft after the month's sinking was 955 feet. Size of shaft cut to let in timber is 23 feet by 8 feet. The drilling was by hand, averaging thirty boys a shift and one white man.

"The depth of holes drilled was between 3 feet 6 inches and 4 feet 6 inches, some being drilled single-handed and others double-handed, according to depth of hole required and amount of space available. A sink was kept ahead, thus always leaving a free side to break to; an average of fifty-eight buckets of débris was hoisted after each blast. Thirty-three cases of dynamite were used during the month. Only sufficient water is let down to keep the timber wet, the rest being picked up at the 250 foot station. With both drums of the engine in gear, and patent detaching safety hooks on the rope, three buckets were used whilst hoisting ground, thus always leaving one in the bottom filling whilst the others were riding.

"When the shaft was 50 feet below the timber, six hitches were cut for bearers; the shaft having been sunk another 20 feet, a light stage was built, the bearers put in, and a set placed in position. This set was used as a staging for the next set, and they were thus built up till they met the others. The sets being 5 feet apart, ten sets were put in at each timbering. This occupied sixteen hours with bearers, dividers and runners, work in the bottom being stopped meantime, so that for six shifts, of eight hours each, during the month no sinking was done."

(3) The third method of placing the holes in the bottom of a shaft is to take advantage of the lay of the formation. This was done in the Robinson Deep shafts,

where the quartzite was dipping about 36° to the south. The general arrangement of the holes in the smaller shaft is shown in Fig. 49.

The excavation is about 18½ feet long and 8 feet wide.

A row of holes, numbers 8 to 14, each about 3 feet deep, is placed along the south side of the shaft; these, in blasting, tend to break away the wedge-shaped piece, shaded in the section in Fig. 49, which separates along a plane of bedding; then other seven holes, numbers 1 to 7, are placed along the north side, and these break away the bulk of the remaining

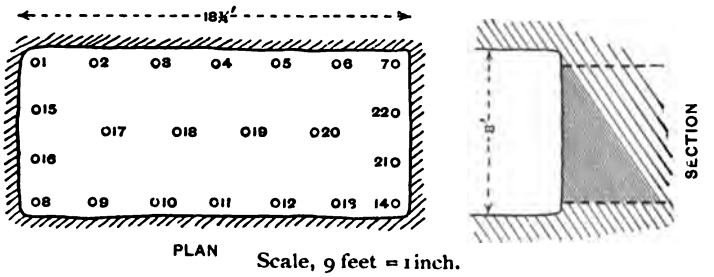


FIG. 49.—Diagrammatic plan and section showing the method of placing the holes in the Robinson Deep.

rock. In further squaring up, eight more holes are necessary, chiefly at the ends, making in all twenty-two holes.

In the twenty-four hours there were three shifts of eight hours each, and on an average there were twenty-four boys working down the shaft each shift; both double-handed and single-handed drilling was used.

The following is an account of the method of sinking which is in use at the Ferreira Deep, from information supplied me by the manager, Mr. Richards.

The size of the excavation is 24 feet by 8 feet, and ten holes make the cut, which is a longitudinal one, as in the Robinson Deep; the disposition of these holes over the bottom is shown in Fig. 50.

The lower cut holes, numbers 1 to 7, are made deeper than any other holes. After the cut has been blasted, fourteen or sixteen other holes are put in to square up, the number

depending upon the way the cut has broken. Blasting is done always about fifteen minutes before the end of the shift, and the down-going shift takes another fifteen minutes to get everything ready to go down, to allow the smoke to clear away.

The rate of sinking in these shafts is about 3 feet every twenty-four hours, and one case of gelatine, containing 50 lbs. nett weight, lasts for two days.

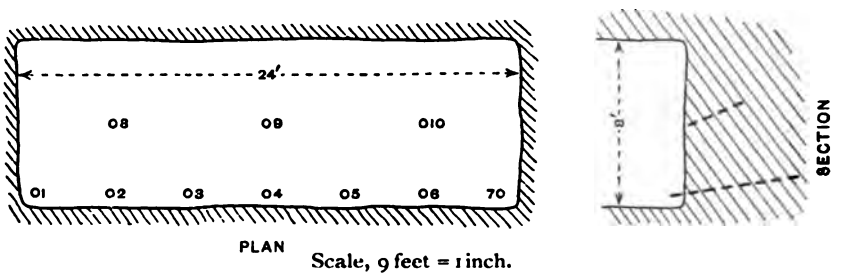


FIG. 50.—Diagram showing the position of the cut holes in sinking the Ferreira Deep shafts.

Sinking an Incline Shaft.—In sinking an incline shaft with hand drilling, advantage is generally taken of the bedding planes to make a cut along the bottom of the face by a series of holes placed in an almost upright position. Afterwards the rock higher up is taken down by holes drilled in a direction less and less inclined to the dip of the formation, till the top holes are placed with the formation. This method of sinking is illustrated in Fig. 51. It is always so arranged that, after the holes have been blasted, a series of benches are left, similar to those which previously existed, so that the bottom of the incline shaft is advanced by stoping the face.

With machine drills a cut is taken out of the centre of the face in the usual way, and afterwards the benches on either side are drilled over and blasted.

The New Primrose Main Incline shaft, an excavation of 17 feet in length and 6½ feet in width, was sunk during the month of October 1894 a depth of 165½ feet with machine drills. Out of a total of 744 hours, 315 were occupied in drilling and blasting, and the remainder, with the

exception of some few hours' delay, in cleaning up; the number of blasts made was thirty-five, each consisting of from twenty-three to twenty-eight holes, each of an average depth of 6 feet; forty cases of gelatine were consumed during the month.

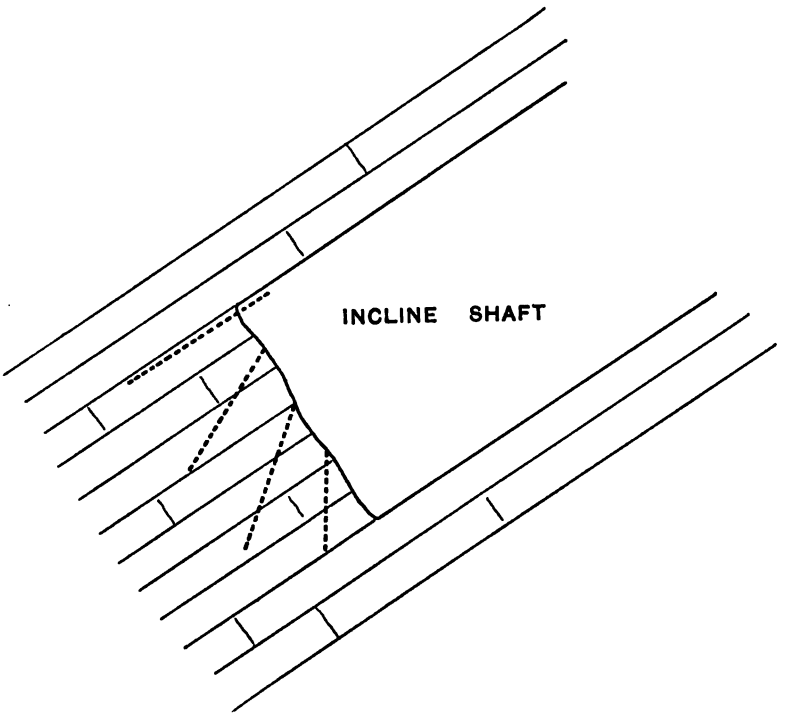


FIG. 51.—Diagram showing the arrangement of holes in the bottom of an incline shaft when drilling is done by hand labour.

Timbering the Shaft.—The shaft excavation is timbered or supported by means of timbers arranged in frames known as "sets."

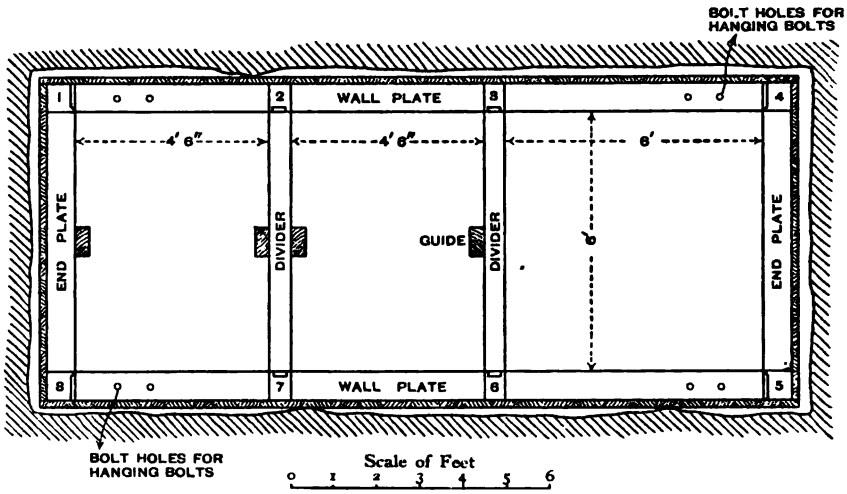
A complete set is shown in Fig. 52; it consists of two "wall plates," which are placed, one on either side, along the shaft; of two "end plates," which are placed across the shaft at either end; and of "dividers," which, by their number and position, divide the length of the shaft into the requisite number of compartments.

These sets are placed at regular distances apart, and

between them posts or "studdles" are placed vertically at each corner and at all points along the wall plates at which the dividers are held.

This framework is boxed in, wherever necessary, by driving planks all round on the outside, and this part of the timbering is termed "lagging."

Wall Plates and End Plates.—These are the principal timbers of the set. They are generally of Oregon or pitch



The areas 1 to 8 are mortises, ½ inch deep, to receive the studdles.

Wall plates 8 × 8 inches	Dividers 8 × 6 inches.
End plates 8 × 8 inches	Studdles 1, 4, 5, 8 : 8 × 8 inches.
Guides 8 × 4 inches	Studdles 2, 3, 6, 7 : 8 × 6 inches.
Hanging bolts ⅞ inch. diam.	
Lagging, planks 9 × 2 inches.	

FIG. 52.—Plan of timber set, framing the shaft excavation.

pine, 8 × 8 inches in section, or, as in the deeper levels, of Kauri wood, 8 × 6 inches in section. In the Robinson Deep they are of the latter wood, but only 6 × 6 inches, though nearer the surface some pitch pine of larger section was used.

The end plates always rest on the wall plates. The joints at the four corners are of the form shown in Fig. 53. These joints are further shown as checked half an inch on top and bottom to receive the upper and under studdles.

When the timber used is in section 8 × 6 inches, the larger dimension is sometimes placed flat in order to oppose its greatest resistance to lateral thrusts, against which the shaft has principally to be protected. As, however, the endlong

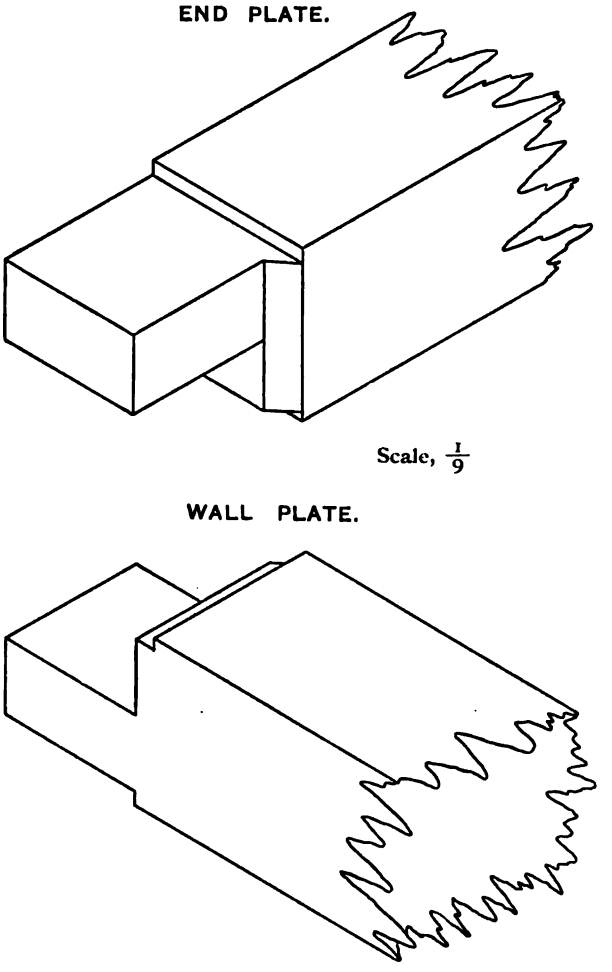


FIG. 53.—Isometric projection of the joint between end and wall plates.

strength of the dividers is powerful to resist such thrusts, the plates may be placed on end, with the advantage that ordinary-sized studdles, 6 × 6 inches, are then flush with the outside of the sets, as shown in Figs. 54 and 55, so that, by transmitting some pressure directly to the dividers, they

relieve the plates to that extent; moreover, less space in the shaft is then taken up by the timber.

In working these plates to shape from the template, the measurements are always taken from the top side, so that if the timber is larger than its standard size, the under studdles are let in more than the normal amount.

Through each wall plate, bolts, called "hanging bolts," are passed, and each set is suspended from the one above by a separate set of bolts; further mention of these is made on p. 173 in connection with fixing and supporting the timber.

Dividers.—The dividers are placed across the shaft from one wall plate to the other, so that the length of the shaft is divided into the necessary compartments. They carry all the guides for the hoisting equipment and have to bear all the strains consequent upon hoisting; in addition, as stated before, they oppose their endlong strength to lateral thrusts.

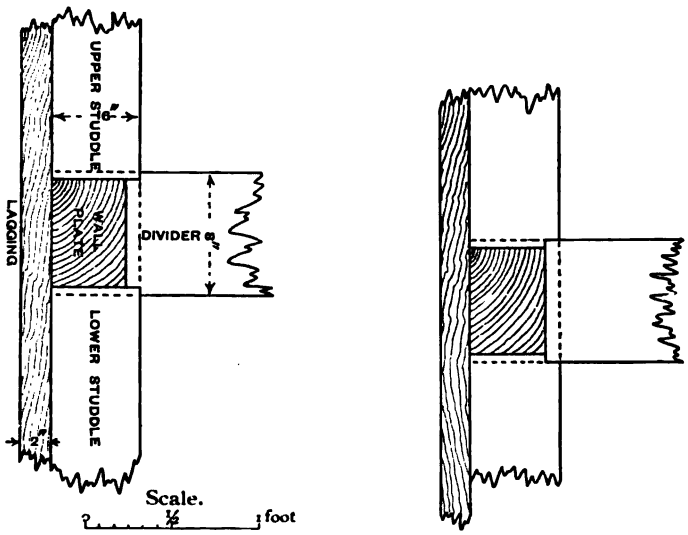
In size they have generally the same depth as the plates and the same width as the studdles, except the smaller one which divides the ladder-way from the pump-way and which generally has only half the width of the others, as shown in Fig. 68.

Figs. 54 and 55 show two methods of jointing the dividers with the wall plates and studdles; they are further held in position by a tapered tenon on either end, generally an inch deep, which is held by cheeks in the wall plates.

Three ways of cutting this tenon are shown in Fig. 56. Of these, No. 1 is in most general use; with it the square end of the studdle bears directly on the top of the tenon, as shown in Fig. 54. This is also the case with No. 3, but in addition that tenon is dovetailed, so that it cannot be drawn out of its cheeks when the studdle is fixed. With No. 2 the studdle bears directly on the top of the dividers, as shown in Fig. 55, so that if it (the studdle) is let into the wall plate, as is usually the case, it must be checked a bit to hold the divider.

As a rule the depth of the divider is well able to spare half an inch top and bottom for the tenons.

The dividers, in most of the shafts which are superintended by the engineering staff of the Consolidated Goldfields, have a deeper section when in position than the wall plates, the measurements being 10 inches and 6 inches respectively; the dividers are then placed so that they extend 2 inches above and below the wall plates. This practice, when used in conjunction with studdles which on the



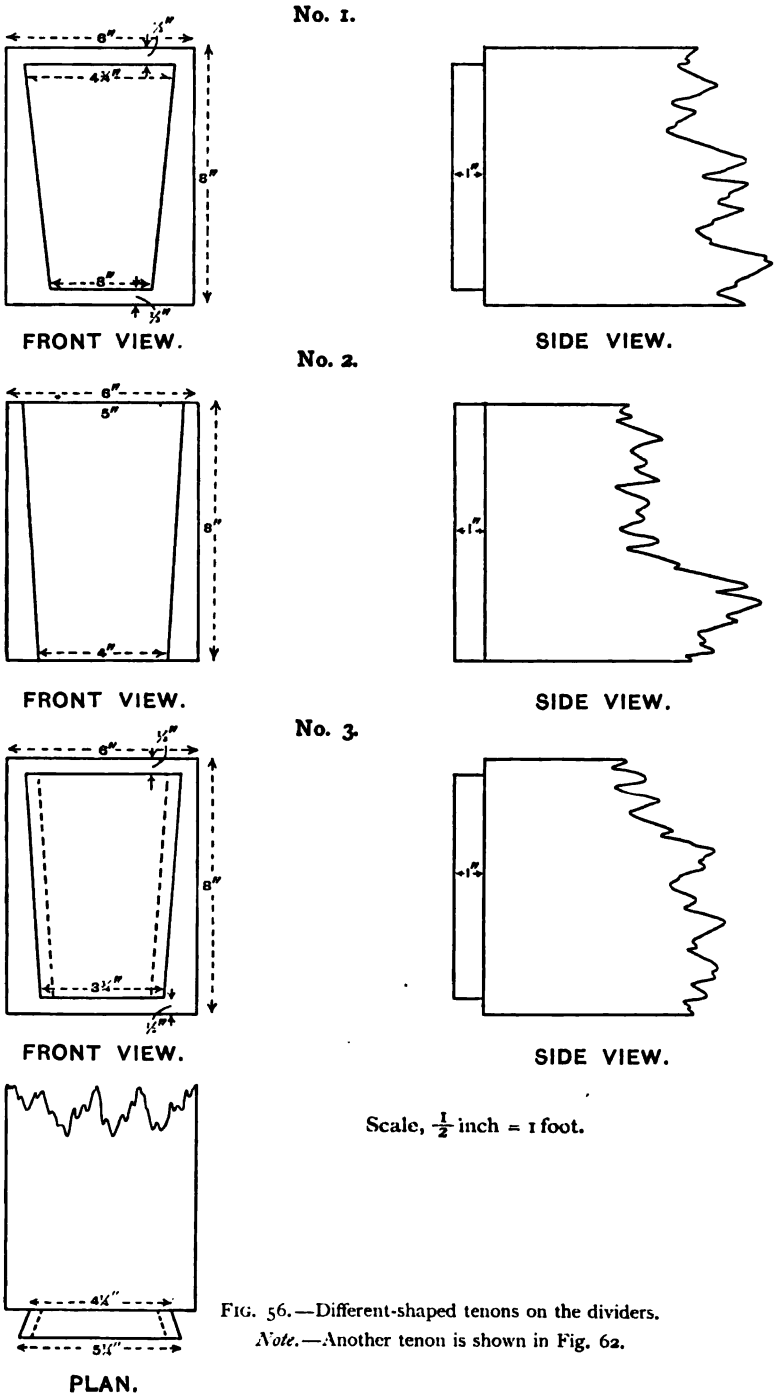
FIGS. 54 and 55.—Sections through divider joints. The dotted lines indicate the true section of the wall plate which has been mortised to receive the studdles and divider.

Note.—Compare the relative position of the lagging and studdle in these figures with that in Fig. 58.

outside are flush with the plates, is good, because the lateral pressure taken by the studdles is transmitted to shoulders of good size on the dividers against which the studdles rest.

It is to the dividers that the wooden guides for running are secured.

Guides.—The guides, which are embraced by the runners of the skip, are made of hard well-seasoned wood, which has been planed and smoothed. The usual size is 4 inches by 5 inches in section, though lately in the deep levels a section



Scale, $\frac{1}{2}$ inch = 1 foot.

FIG. 56.—Different-shaped tenons on the dividers.
Note.—Another tenon is shown in Fig. 62.

of 4 inches by 8 inches has been put in. They are attached by coach screws, 8 inches long, to the dividers, either directly, or with distance pieces between, as shown in Fig. 57.

The disadvantage of having distance pieces interposed is that, though the guides may be set more truly in line throughout the shaft, the attachment is not sufficiently firm for fast winding.

The advantage of having guides of the larger size is that

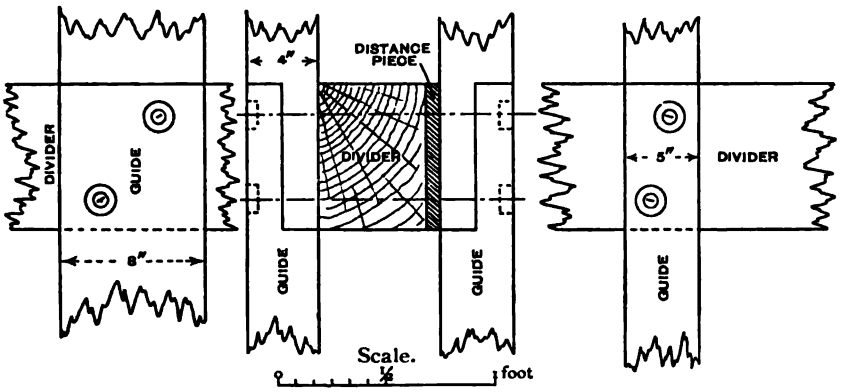


FIG. 57.—Showing the attachment of guide to divider with and without a distance piece.

the screws get a firmer hold upon the dividers, and thus greater rigidity is obtained.

The guides are usually made in such lengths that a joint between two lengths may occur at a divider.

Studdles.—These are the upright posts which stand vertically between the sets; they are placed at the corners and on the wall plates at those points where the dividers are joined to them. They serve to support the sets upwards from the bearers, to keep the dividers from rising, and to distribute any lateral pressure.

The general size of timber used is 6 × 6 inches square, and the length depends on the distance the sets are apart, 5½ feet being a usual length.

Where the plates are 8 inches wide, on their flat, and the studdles 6 × 6 inches, the latter are placed flush with the

inside of the plates only, as shown in Fig. 58. It is better to arrange that in addition they are flush with the outside, as shown in Figs. 54 and 55, for then the whole structure can be well boxed in with lagging and any pressure on the sides will be distributed; this is done by having one sectional dimension of the studdle equal to the width of the plate laid flat.

In the Vogelstruis Deep the corner studdles are of the same size as the plates, viz. 8 × 8 inches, and the intermediate ones are 8 × 6 inches.

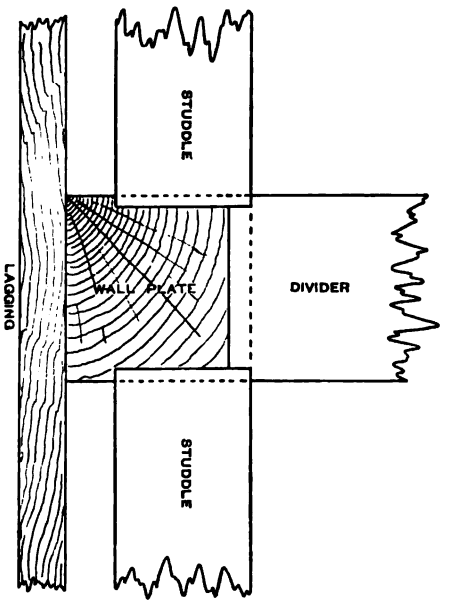


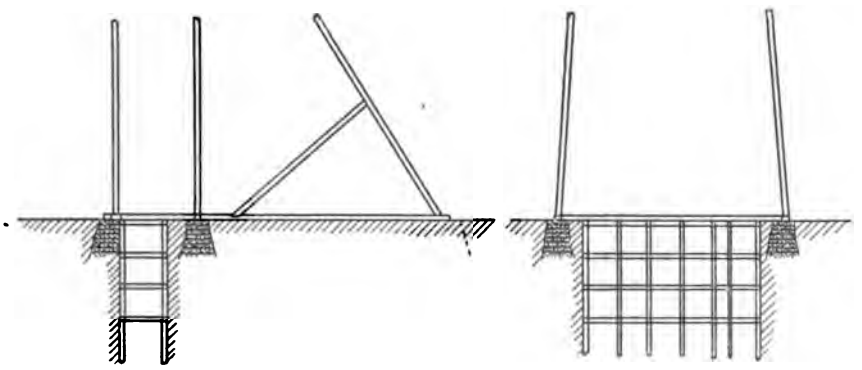
FIG. 58.—Diagram showing the studdle not flush with the outside of the set.

It is the usual practice to insert the square end of the studdle into a shallow mortise, about half an inch deep, so that the intermediate studdles bear on the tenons of the divider, as shown in Fig. 54.

As stated before, with the tenon marked No. 2 in Fig. 56, it is necessary to cut the studdle at the end and on the inside, as shown in Fig. 55, in order to cover the divider. Studdles are rarely mortised in, because the resistance to great lateral pressure, which would be afforded by the area at the root of the tenon, is less than that which would be afforded by the whole side of the studdle against a shoulder, and also because it is not possible to work the timber so true that in opposition to crushing pressure the joint would be bearing equally on the shoulder and on the end of the tenon. In Fig. 62, however, the studdles of the timbering of the Main Incline shaft of the Durban-Roodepoort are shown with mortised ends.

Lagging.—Planks 9 or 10 inches wide are used for lagging. Where the ground is bad, or just near the surface, they are 2 inches or sometimes 3 inches thick; in better ground they are 1 inch thick, and in hard quartzite no lagging is used at all.

Fixing and Supporting the Timbers.—The timbering is generally started from the surface after the excavation has reached a depth of about 40 feet. Around the mouth of the shaft a platform of concrete, filled in between masonry walls, is constructed, and on it the "collar set" of the timbering is laid. This consists of two wall plates, which extend on either



Scale, 30 feet = 1 inch.

FIG. 59.—Diagram of headgear foundations of the five compartment shaft of the Ferreira Deep, Limited.

side of the excavation for about 4 or 5 feet, and rest on two other timbers placed crosswise at either end of the excavation; all of these timbers are carefully levelled and well bedded.

The method of finishing off the shaft top, and of putting in the foundations to carry the headgear, is shown in the accompanying section of the shaft of the Ferreira Deep, Limited (Fig. 59).

From the collar set the others are suspended, each from the one above it, by bolts called "hanging bolts," through the wall plates, until the weight is taken upon other timbers called "bearers," which are placed across the ends of

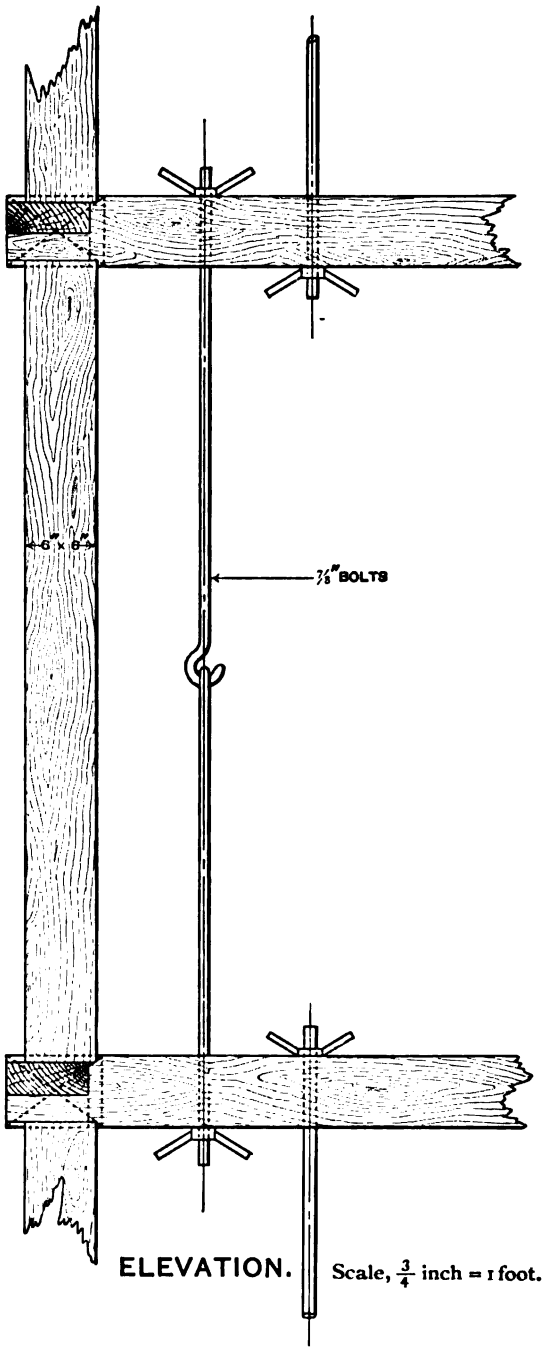


FIG. 60.—Diagram showing the method of fixing hanging bolts between two sets.

the shaft and hitched securely into solid rock; when this has been done the bolts have served their purpose and the sets are supported from below, on the studdles.

Hanging Bolts.—These are seven-eighths of an inch or an inch in diameter; they are generally made with a hook at one end and a thread at the other, and of such a length that they extend half the distance between two sets, and then hook one into another. This arrangement is shown in Fig. 60; the advantages are that the wall plates with these bolts through them can be more easily swung or hooked into position, and again, if either half should get bent, it is easily taken out. In some mines the bolts extend the whole distance between two sets, and in these cases it is necessary to swing the plate into position with the help of extra eye-bolts and chains of about the same length.

The number of bolts between two sets depends upon the size of the shaft. In a small shaft two bolts in each wall plate, one near either end, are sufficient, but in a large shaft three are used, one at either end and another in the centre. No bolts are put through the end plates.

For every line of bolts in a shaft there are two holes in a wall plate of each set, one of which is for connection with the set above, and the other with that below. These holes are placed from 6 to 9 inches apart, and they are arranged so that the bolts between alternate sets are in line, as is shown in Fig. 60.

Ordinary nuts with good stiff washers are more often used for drawing the sets together than the wing-nuts shown in Fig. 60.

Bearers.—These are stout pieces of timber placed across the shaft and supported at either end on solid rock, as shown in Fig. 65.

If possible they are placed where the solid rock offers a good shoulder for the hitch. This latter is either chiselled out, in which case it is sufficient that a ledge of 3 or 4

inches is obtained, or more often it is made by careful blasting with small charges, and in these cases it is necessary to have a larger hold, some 18 inches or 2 feet, for the timber. Another method is to drill a series of holes completely around the border of the required hitch, and then to break the enclosed rock by means of a small "pop shot" centrally placed.

A hitch having been made on either side, the stout timbers forming the bearer are wedged firmly into position.

Shaft timbering is supported at points from 75 to 100 feet apart by placing such bearers underneath the end pieces, and if necessary under the dividers also. These bearers consist usually of two pieces of stout timber placed one on the other, making a depth of about 18 inches. They are made to take up the weight by wedging.

In the Vogelstruis Deep, in addition to the wooden bearers under the end plates, the wall plates are carried on iron bracket-bearers such as are illustrated in Fig. 61.

The wall plate rests on the horizontal limb, 18 inches of which, near the end, are rounded off for insertion into a drilled hole, whilst the end of the lower limb rests securely on a small hitch.

The procedure in timbering may be briefly described as follows: The wall plate is sent down, attached by chains and eye-bolts from its centre bolt-hole to the bottom of the sinking bucket. Arrived at the proper level, the lashing by which the plate was kept upright for lowering is loosened, and the plate comes down to the horizontal position, being balanced around its centre bolt-hole. The hanging bolts are then placed through the end holes, and the plate is slung over so that the bolts through it may be hooked on to those which hang down from the last set; the chain and eye-bolt through the centre are then released. The other pieces are brought down in the bucket and placed in position. After the set has been completed it is fixed by careful wedging and blocking. The wedges are about

10 inches long, 4 inches wide, and they taper from $1\frac{1}{2}$ inches at the thick end to an edge at the other. They are used in conjunction with blocks, and they are always driven downwards, so that if they become loose they cannot drop

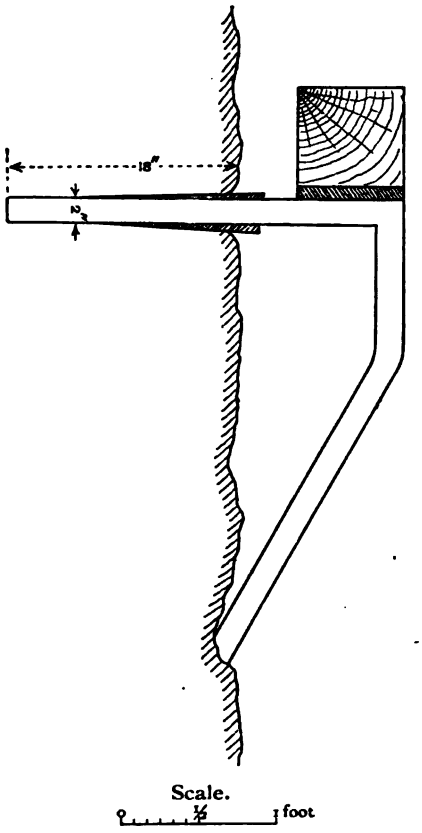


FIG. 61.—Iron bearer for wall plates used at the Vogelstruis Deep.

out. The blocks are always placed in line with the principal timbers.

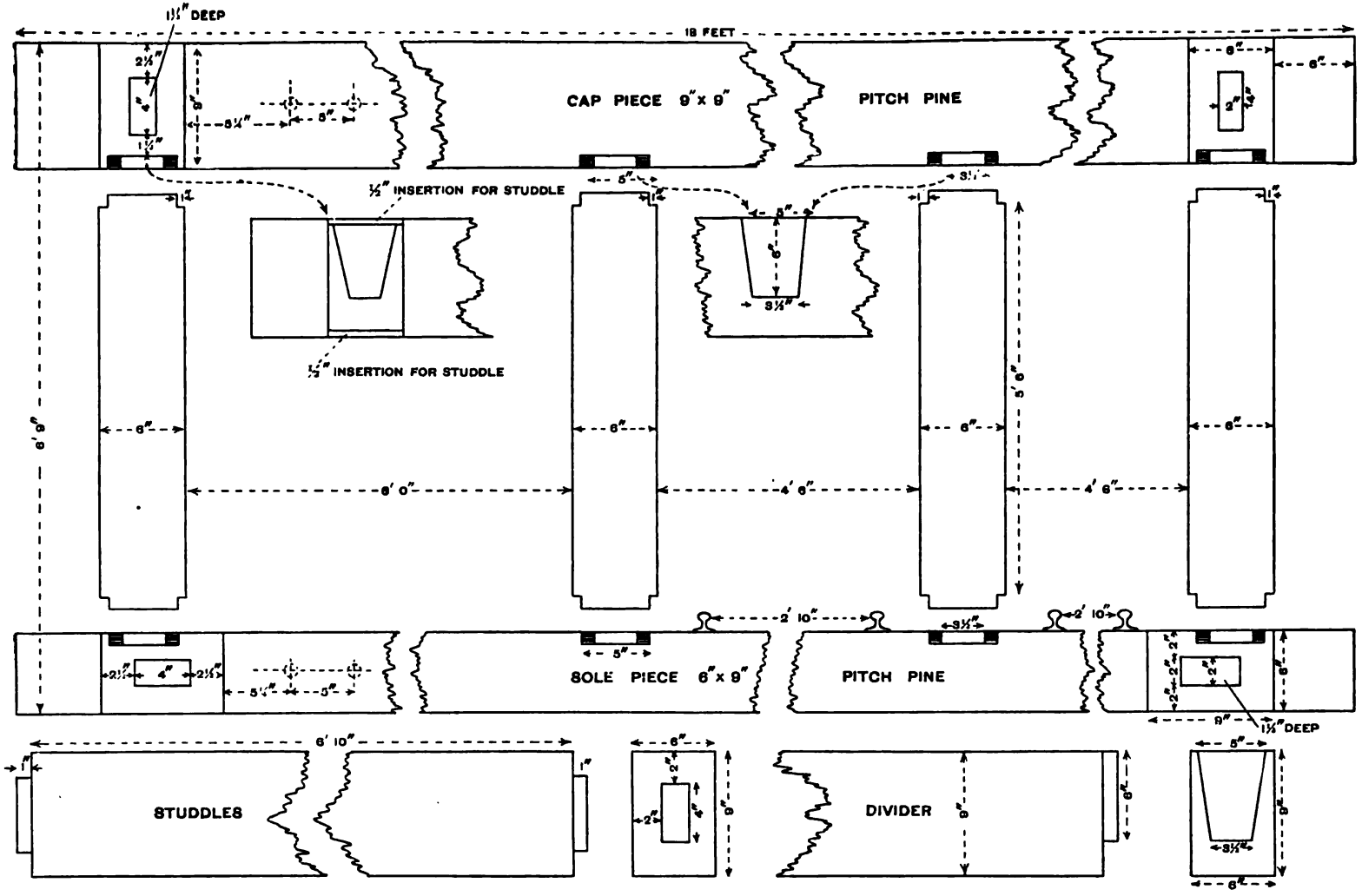
In the Central Roodepoort an entirely different procedure is followed. Bearers of 16 inches by 9 inches are put in about every 40 feet under the end plates, and at intervals of 80 feet other bearers of 12 inches by 6 inches are placed under the centre divider. From these bearers the timbers are built up so that no hanging bolts are required.

There are, however, for greater security, two lines of bolts down the shaft, and another bolt, which passes near the centre divider from one wall plate to the other, is used for bracing the set before blocking. On top of the bearers heavy sets of 8 inches by 9 inches in section are placed, and the lower sides of these are protected from the effects of blasting in the shaft bottom by iron plates $\frac{5}{8}$ inch thick. If the bearer set is not just in the proper place for a set, false or short studdles are placed on it, and on these the next set is erected in conformity with the correct distance. This method of timbering is also in use at the Angelo Deep, as described on p. 162.

Incline Shaft Timbering.—All incline shafts are timbered near the surface and down to solid rock with sets in which all the separate timbers which are used in vertical timbering are represented.

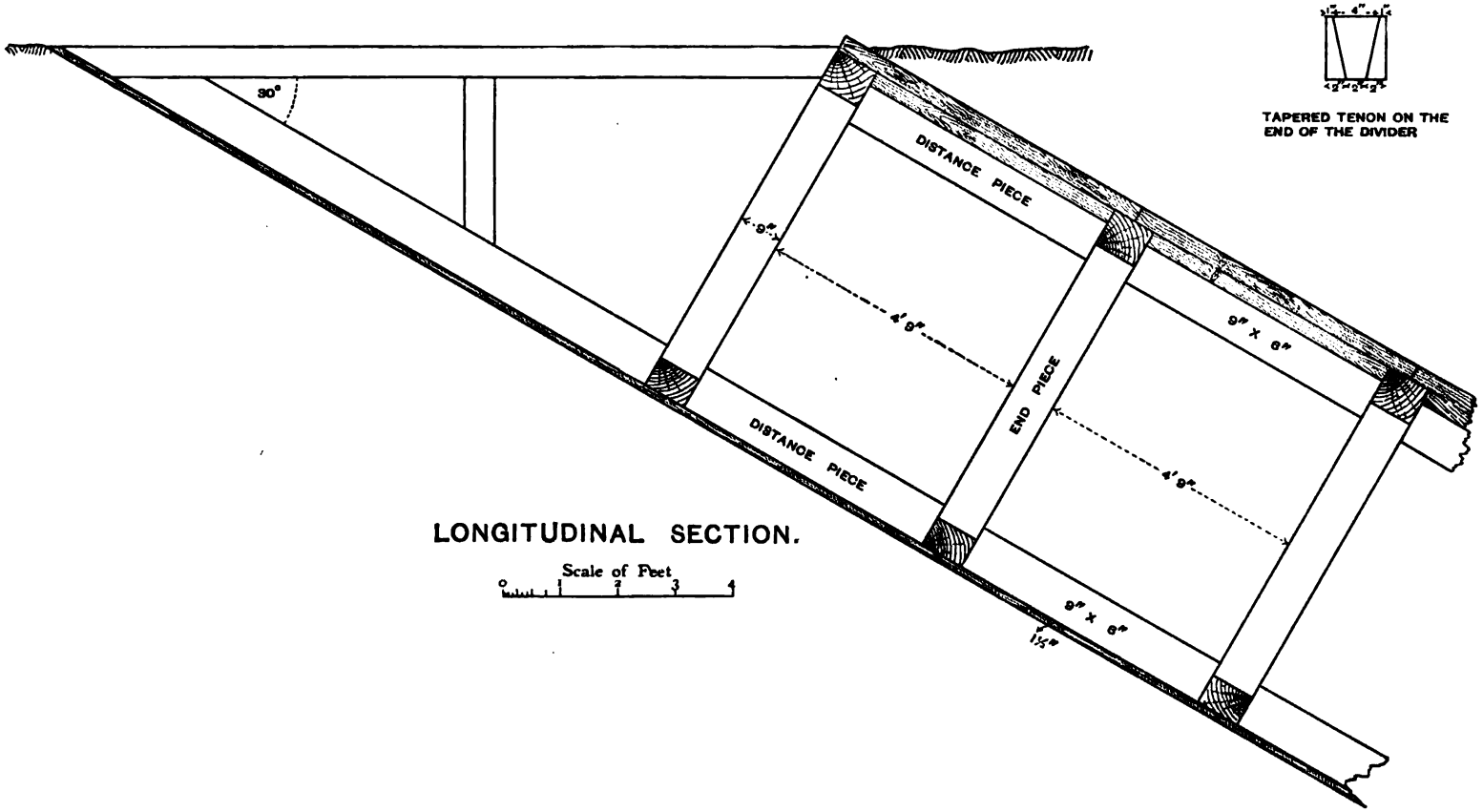
Complete incline sets are shown in Figs. 62 and 63, the first being that of the Durban-Roodepoort Main shaft, which dips about 31° throughout, and the second, that of the City and Suburban Main Shaft, which dips 30°. In this latter the mouth of the shaft is also shown. The sets are placed, near the surface, with their centres from $4\frac{1}{2}$ to $5\frac{1}{2}$ feet apart. Lower down, and in more solid rock, this distance is determined to some extent by the necessity of having sole pieces as sleepers for the hauling track, and so they are not often much more than 6 feet apart.

The cap piece and the sole piece or sill correspond to the two wall plates. The former is generally thicker along the plane of the set than the sole piece, especially in heavy ground, but of the same depth. They are both mortised to receive the tapered tenons of the dividers, and in some cases, as in the Durban-Roodepoort, those of the end pieces as well. They are also mortised to receive either the square end or the tenon of the studdle, which in inclined timbering is more often known as the "distance piece." In the Durban-Roode-



Scale, 1 inch = 1 foot.

FIG. 62.—Timbering of the Durban-Roodepoort Main Incline shaft.



LONGITUDINAL SECTION.

Scale of Feet
0 1 2 3 4

TAPERED TENON ON THE END OF THE DIVIDER

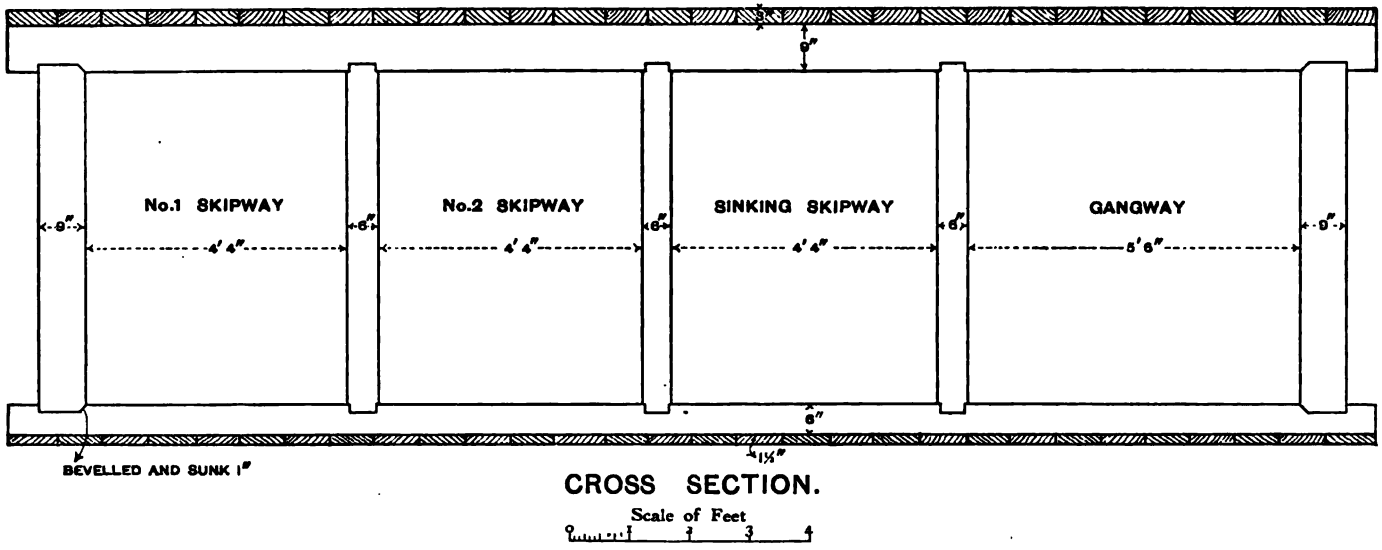


FIG. 63.—Timbering of City and Suburban Main Incline shaft.

poort set, Fig. 62, this mortising of shaft timbers is well shown.

The more usual joint of the cap and sole with the end pieces is made by checking the former to a depth of about an inch, with a bevel on the inner side for a shoulder to hold any pressure from the outside. This joint, called a "clap-me-down" joint, is shown in the City and Suburban set, and also in Fig. 64.

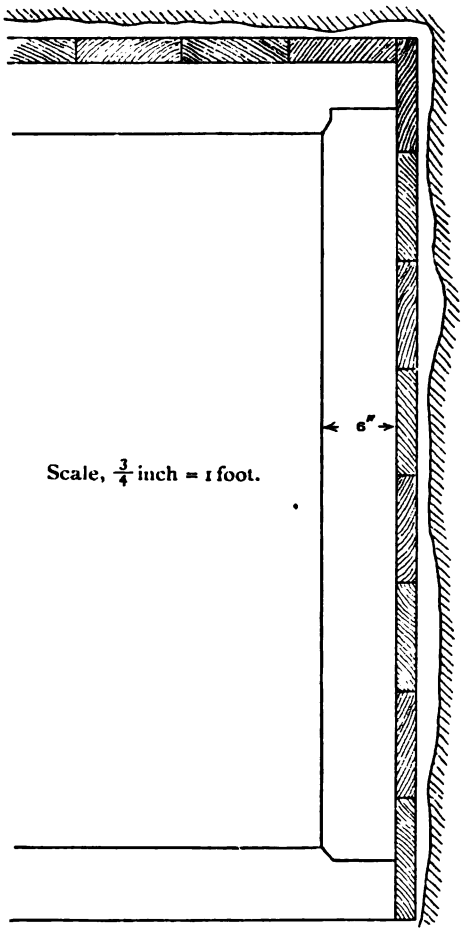


FIG. 64.—Clap-me-down joint between cap and sole pieces.

The cap and sole pieces generally extend beyond the end pieces, as shown in Figs. 62 and 63, so that these latter are secured from falling towards the outside, but it is generally considered that with lagging, blocking and packing there is little chance of any pieces falling out that way, and that the joint shown in Fig. 64 is a good one. The end pieces are further secured by the distance pieces. The dividers have principally to support the cap piece. They generally have the same depth as the main timbers, but the other sectional dimension is less. In the City and Suburban, however, they are smaller all round. The tenon used in that mine is similar to No. 2, Fig. 56, and it has 2 inches of taper. In the Durban-Roodepoort the tenon

bears against three cheeks of the mortise, as shown in Fig. 62.

The distance pieces are subject to little strain in a shaft at an angle of about 30°, so they are the least important of the timbers of an incline set. In the Robinson West Incline there are only four distance pieces, one in each corner. They are of 4 × 3 inches section, and are nailed in position. In the City and Suburban, which is a larger shaft, there are six, placed one in each corner and one above and below the centre divider; these are 6 × 9 inches in section. The lower three are placed flush with the bottom of the set, but the upper ones are flush with neither the inside nor the outside of the framing.

In the Durban-Roodepoort there are four large distance pieces, one in each corner, placed to lie flush with both the inside and the outside of the framing.

Lagging is made from planks 9 or 10 inches wide and from 1½ to 3 inches thick. It is placed more frequently over the top and around the sides of the shaft than under the sole pieces. In the City and Suburban extra lagging is packed between the top of the distance pieces and the lagging proper.

Hanging bolts exactly similar to those shown in Fig. 60 are sometimes, though not invariably, used.

In the solid rock the cap and end pieces and some of the dividers are not used; the sills, and that divider which separates the pump and ladder way from the hauling ways, alone are necessary. In order to secure these, the sill is secured in a hitch on either side, and the divider, with its lower end pressing on the sill, is firmly wedged against the roof of the shaft.

Incline timbering is supported by a collar set at the surface, and by bearers placed at regular distances underground, which are in every way identical with those used in vertical shafts. It is, however, not necessary to have as many bearers.

Guides are not used in inclined shafts, their place being taken by rails secured to the sole piece, as will be more fully described in Chapter XI.

Timbering of the Angle Connection.—Where a vertical shaft turns off on to an incline, special timbering is generally used. Fig. 65 shows the structure at the Jumpers Deep.

The incline on to which the shaft is turned is one of $28\frac{1}{2}°$ from the horizontal, and the radius of the curve of the winding track is 50 feet. The weight of the vertical timbering is taken by a bearer of 16 × 6 inches section, which is carried in hitches just above the commencement of the curve (Fig. 65).

All the loose rock is taken from the top of the wedge-shaped piece of ground between the two shafts, and cemented masonry is built up in its place to form a solid bed for the sole piece of the angle connection. This sole piece is 12 × 6 inches in section, and it extends in one piece across the vertical shaft to be supported in a hitch on the other side. The line of the sole piece is also extended across the incline shaft by another piece, which rests on the top end of a bearer fixed in the incline shaft. On the sole piece two main uprights, 12 × 6 inches in section, are seated in line under the wall plates of the vertical shaft timbering; these extend, in one piece, right up to the bearer above, a length of about 26 feet.

There are two other upright pieces arranged as steps; the upper one is seated about half-way up on a cross piece and at a distance of 3 feet centres, towards the incline, from the main upright; the lower one is seated on the masonry, and is placed at a farther distance of 3 feet centres towards the incline.

Into these uprights horizontal pieces of 12 × 6 inches section are mortised, at distances of $4\frac{1}{2}$ feet centres apart. The top one is equal in length to the width of the vertical shaft, and that one which is above the sole piece covers both vertical and incline shafts. The whole structure is bound together by bolts, and it is further held by blocking against the solid rock.

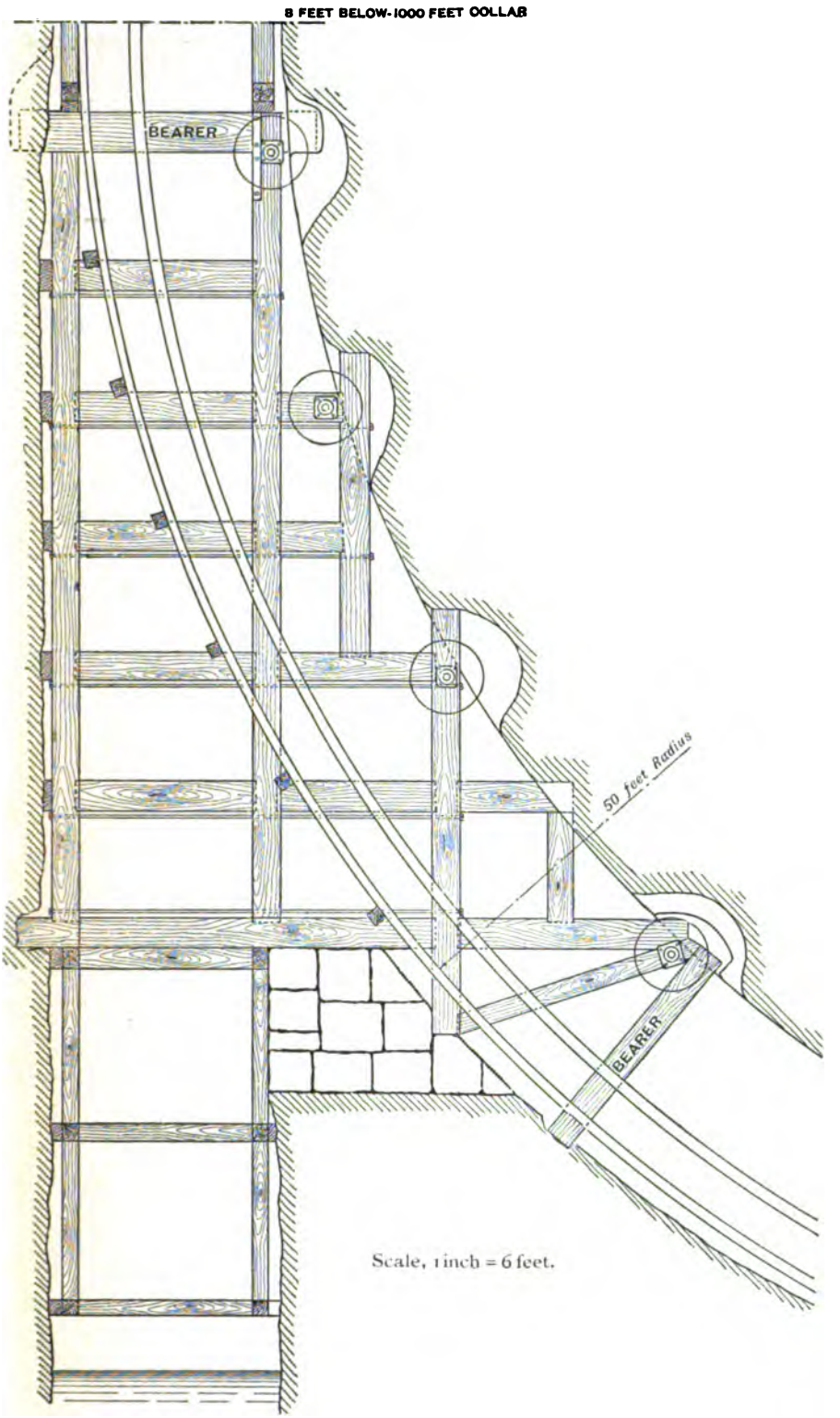
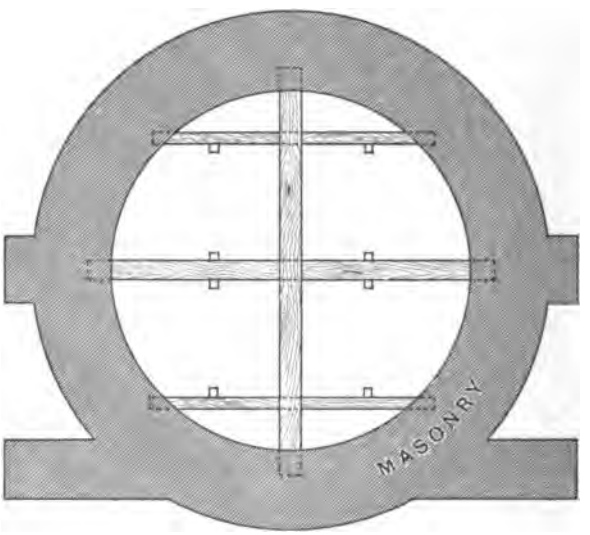


FIG. 65.—Shaft timbering at the angle connection between a vertical and an incline shaft, for continuous hoisting.

On the horizontal pieces are placed the sleepers, to which the rails that form the guides for hoisting are attached. There are four pulleys arranged to guide the rope round the curve. Where the angle on to which the shaft is turned is much higher there is no need for such a structure, as the timber can be taken round by making the studdles on the outside of the curve larger than those on the inside. This, however, can only be done where the curve has a large radius, say 150 feet.

For the Hammond shaft of the Simmer and Jack East, Mr. R. M. Catlin has substituted a parabolic curve for the circular one usually employed. This is said to admit of a far higher speed of winding than is safe with the ordinary form. The method of putting in the timbering for this curve is shown in Fig. 67, for which I am indebted to Mr. J. B. Pitchford.

Circular Shaft.—In all the gold mines of the Witwaters-



Scale, $\frac{1}{8}$ inch = 1 foot.

FIG. 66.—Plan of the mouth of the Langlaagte Royal circular shaft.

rand there are only two circular shafts—one at the New Primrose, 11 feet in diameter, inside of the walling, and 400 feet

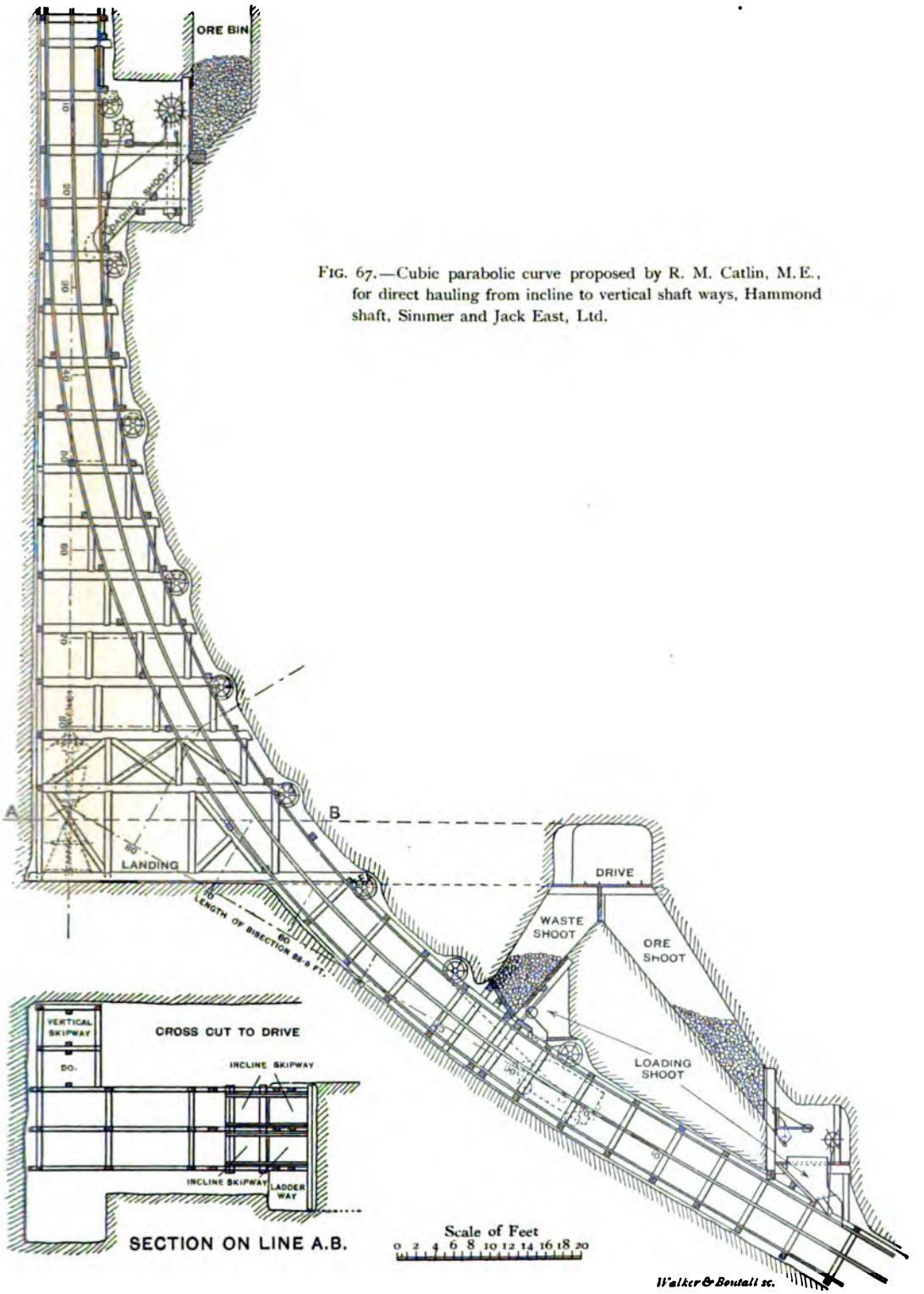


FIG. 67.—Cubic parabolic curve proposed by R. M. Catlin, M.E., for direct hauling from incline to vertical shaft ways, Hammond shaft, Simmer and Jack East, Ltd.

deep; and the other at the Langlaagte Royal, 15 feet in diameter, and laid out to go 900 feet in depth.

A plan of the mouth of this latter is shown in Fig. 66. The guide sets or dividers are, near the surface, about 9 feet apart from centre to centre; the two main timbers at right angles are 10 × 8 inches in section, and the smaller timbers are 6 × 8 inches; there are four winding compartments, as shown, and the skips or cages run on wooden guides fixed to the dividers; the pump columns and air mains are placed out of the way around the circumference.

Sumps, Lodges.—At certain places in a vertical shaft, lodges or sumps are cut to collect or hold the water, and to provide a chamber from which the water may be pumped.

It is usually arranged to catch the surface water at a depth of about 150 feet, and to pump it separately to the surface, because, being clean, it can be fed into the boilers, whilst water from the shaft bottom is unsuitable for this purpose. This is done by cutting the shaft so that a shoulder, as though for a hitch, is made all round; on the edge of this a line of timber is placed, and so bedded in cement or clay that a channel for the water is formed inside it. Into this channel the water coming down the shaft is guided by short planks placed in an inclined position to intercept its fall, and led into the sump.

The shaft is ringed in this way wherever there is a sump, or wherever the water coming into the shaft is heavy; it is also usually done just above each station to shelter those employed in onsetting.

Sumps, with the exception of that one which holds the surface water, are placed to suit the throw of the pump. In a good many of the deep mines electrical pumps, capable of forcing the water 500 feet vertically, are to be used in series, so that the sumps will be placed at intervals of 500 feet; for these pumps it is also necessary to cut out a lodgment, though the Cornish pump does not require any. These

spaces are cut out at the pump end of the shaft, an excavation about 8 × 8 × 9 feet being required for the pump, and one about 14 × 15 × 9 feet for the sump, though the size of this latter must be proportionate to the amount of water coming; it is usually so arranged that a sump will take some few hours to fill when the pumps have stopped.

Rate of Sinking and Cost of Shafts.—The cost of shafts includes that of sinking and timbering and that incurred in cutting the necessary sumps and stations for the pumping system, in addition to that of actual sinking.

The following are two statements of the cost of sinking and timbering 89 feet of the Roodepoort Central Deep shaft during April 1897, from information kindly supplied by Mr. H. B. White, the manager.

No. 1 shows the distribution of the cost under the heads of the different operations, and No. 2 under those of the various component items.

No. 1.

	Total Cost.			Cost per Foot.			Percentage of Cost.
	£	s.	d.	£	s.	d.	
Sinking	646	18	0	7	5	4.5	48.3
Drill sharpening	37	5	0	0	8	4.4	2.8
Timbering sets	244	15	9	2	15	0.0	18.3
" ladder-landings	24	19	5	0	5	7.3	1.9
" bearers	7	15	9	0	1	9.0	0.6
Hoisting	121	17	6	1	7	4.7	9.1
Pumping	67	0	0	0	15	7.0	5.0
Shaft top expenses	73	0	0	0	16	4.5	5.5
Management and salaries	114	3	4	1	5	8.0	8.5
Totals	£1337	14	9	£15	0	7.4	100.0

No. 2.

	Total Cost.			Percentage of Cost.
	£	s.	d.	
White labour	518	4	6	38.8
Native „	358	8	6	26.8
„ food	51	8	6	3.8
Compound expenses . .	40	0	0	3.0
Timber	156	8	4	11.7
Explosives	101	14	2	7.6
Lubricants	4	10	0	0.4
Miscellaneous stores . .	33	4	3	2.4
Fuel	73	12	6	5.5
Totals . . .	£1337	14	9	100.0

The excavation for the shaft is 23½ feet long and 8 feet wide, the measurements inside the timbers being 22½ feet and 6 feet. The total depth at the end of April was 511½ feet.

When on a property, shaft-sinking alone is proceeding, the whole of the general expenses are charged to it; but where a mine is in the producing stage, these would be distributed over several accounts, and consequently would bear less heavily on the shaft-sinking. Reference to No. 1 statement above shows that the actual cost of sinking, including drill-sharpening, is about one-half of the inclusive cost. In sinking the incline shafts of the outcrop companies, the inclusive cost is generally not much more than that of actual sinking, for little timbering is required, and the larger proportion of the costs under the other items are charged to other accounts.

It is generally found that the complete cost of timbering a five-compartment vertical shaft varies from 65s. to 70s. per foot of shaft.

The following is a tabulated statement showing the costs of some of the shafts along the Rand :—

Cost and Rate of sinking Vertical Shafts.

Name of Mine.	Name or Number of Shaft.	Total Depth.	Size of Shaft Excavation.	Rate of Sinking, Feet per Month.	Cost per Foot of Shaft sunk.	Remarks.
		Feet.	Feet.		£ s. d.	
Rose Deep . . .	1	911	23 × 8	...	31 1 4	
" . . .	2	714	18 × 8	...	26 16 5	
Nourse Deep . .	1	991½	Small	...	15 10 7	
" . .	2	1580	3 compartments	...	20 1 3	
" . . .	3	1483	"	...	20 11 2	
Jumpers Deep . .	1	1057	23 × 8	65	24 9 0	
" . . .	2	1352	23 × 8	...		
Glen Deep . . .	1	1005	18½ × 8	91	22 4 2	
" . . .	2	1017	18½ × 8	66	24 2 10	Sunk in dyke from 42 feet to bottom.
Langlaagte Deep . .	1	1174	23 × 8	60	26 3 5	
" . . .	2	968	23 × 8	60	22 17 7	
Durban-Roodepoort Deep	2	1444	18 × 8	87	19 10 5	
Robinson Deep . .	1	2389	22 × 8	69	25 14 7	Sunk entirely by hand drilling; the costs include an amount for depreciation of plant and machinery used in sinking.
" . . .	2	1923	18½ × 8		24 3 7	
Crown Deep . . .	1	1324	22 × 8	...	33 14 8	
" . . .	2	1100	22 × 8	...		
Vogelstruis Deep . .	East	891	18 × 7½	65	20 14 0	Hard dyke from 250 feet to bottom.
" . . .	Central	1055	18 × 7½	89	18 12 4	
" . . .	West	960	18 × 7½	86	15 5 9	Sunk entirely by hand labour.
West Roodepoort Deep .	1	734	18 × 7	67	19 6 0	
" " .	2	926	18 × 7			
Knights Deep. . .	Robertson	885	28 × 8	55	34 4 3	Sinking interfered with by water.
" . . .	Connor	716	28 × 8	40	41 7 11	
Simmer and Jack East .	Clement	1546	28 × 8	94	22 7 11	Sunk entirely by hand labour.
" " " .	Hammond	1518	28 × 8	96	23 3 3	" "
" " " .	Lohse	691	28 × 8	70	28 12 2	" "

These costs are inclusive, and agree with the figures in the balance-sheets of the various companies concerned.

The Nourse Deep shafts, which cost, inclusive of all charges, £15 : 10 : 7, £20 : 1 : 3, and £20 : 11 : 2 per foot respectively, were sunk on contract prices of £6, £7 : 10s., and £7 : 10s. per foot respectively, the contractors having to provide labour, lights, and explosives, but not timber or timbering.

Reference to this table shows that, where in any one property shafts of different sizes have been sunk, the smaller shafts have cost somewhat less per foot, but not as much less as would be proportionate to their smaller size.

It is also almost the invariable experience that the shaft which has been sunk quicker than others on the same property has cost less per foot.

The costs of the Robinson Deep shafts include a certain amount for depreciation of temporary plant put up for sinking. The high cost of the two shafts of the Knights Deep was due to a large influx of water.

No. 2 shaft, Glen Deep, was sunk in hard dyke from a depth of 42 feet to the bottom at 1017 feet; in consequence, the rate of sinking was less and the cost greater than No. 1 shaft, although that shaft, in addition to the ordinary costs, had to bear the expense of re-timbering a portion.

Similarly, the greater cost and less speed of sinking the east shaft of the Vogelstruis Deep were due to a hard dyke, which was struck at a depth of 250 feet from the surface, and which was still in the shaft at a depth of 891 feet.

The cost of incline shafts is less than that of vertical shafts, chiefly because considerably less timber is required; sinking also proceeds quicker, because when blasting has to be done, everything has only to be drawn up to the level above, and not to the surface, and less time is, in consequence, lost.

The Robinson Main Incline, 15 × 5½ feet in the clear, cost, with rails complete, about £9 per foot.

The contractors at the City and Suburban, working with hand labour, get £8 per foot sunk, the size of the shaft being 20 × 5½ feet in the clear.

The rates of sinking given in the statement on p. 191 were obtained by dividing the total depth sunk in each shaft by the number of months which had lapsed since it was started, so that they include all the ordinary delays, such as cutting stations, influxes of water, etc. No extraordinary delays, such as the entire and prolonged suspension of operations, are, however, included.

On p. 160 the high average rate of 121 feet per month is shown to have been maintained for six months in No. 2 shaft of the Durban-Roodepoort Deep.

On p. 161 a description of the sinking of 156 feet in the Clement shaft of the Simmer and Jack East is given, and on p. 162 a description of 164 feet sunk in one month in one of the Angelo Deep shafts.

On p. 164 mention is made of 165½ feet sunk in a month in the Main Incline shaft of the New Primrose.

All the rates of sinking which are now being made are very great advances on those of two or three years ago, when 50 feet per month was a fair average over an extended period.[1]

Comparison of Machine and Hand Drilling in Sinking.—

[1] Since work was resumed on the Simmer and Jack West, 21st January 1898, and on the Jupiter during January 1898, the following depths have been sunk :—

	Simmer and Jack West (Howard Shaft).	Jupiter (one shaft).
During January	21 feet	68 feet
,, February	80 ,,	122 ,,
,, March	130 ,,	131 ,,
,, April	109 ,,	133 ,,
,, May	183 ,,	169 ,,
,, June	160 ,,	150 ,,

The cost of 183 feet sunk in May in the Howard shaft was at the rate of £15 : 17 : 2 per foot ; the excavation of each shaft was 28 feet long by 8 feet wide.

The following statement of information supplied by the manager, Mr. F. H. P. Cresswell, refers to the No. 2 shaft of the Durban-Roodepoort Deep :—

	Hand Drilling.	Machine Drilling.
Period of sinking . . .	22nd June to 31st December 1895	1st May to November 1896
Depth sunk	450½ feet	760 feet
Average rate, feet per month .	71¾ ,,	121½ ,,
Total depth of shaft at the end of period	450½ ,,	1444 ,,
Cost per foot	£17 5 10	£18 5 4

The percentages of cost, which make up the total of £18 : 5 : 4 for machine drilling, are :—

Wages, white and black . . .	45.7 per cent.
Timber	13.1 ,,
Explosives	10.1 ,,
General stores	4.0 ,,
Hauling and Pumping . . .	11.2 ,,
General	7.0 ,,
Shops (drill-sharpening, etc.) . .	1.5 ,,
Rock drills and compressors . .	7.3 ,,
	99.9

From the percentage of explosives used, it can be calculated that one ton of rock cut required 1.25 lbs. of blasting gelatine.

Of the three shafts of the Vogelstruis Deep, the west shaft, which cost considerably less than the others, was sunk entirely by hand drilling, whereas in the other two, machines were chiefly employed.

In sinking by hand labour the three shafts of the Simmer East to the depths stated in the table on p. 191, two were advanced at a higher average rate than any others along the Rand, and the following percentages of the main items of cost were obtained :—

White labour	27.43	per cent.
Native „	33.00	„
Timber and timbering	15.00	„
Lubricants	0.56	„
Fuel	4.00	„
General charges	7.48	„
Maintenance	0.15	„
Office expenses	5.00	„
Explosives	5.12	„

From the above percentage of explosives it may be calculated that one ton of rock cut required about $\frac{1}{2}$ lb. of gelatine.

In the shaft of the Central Roodepoort Deep—details of the cost of which were given on pp. 189 and 190—where sinking is being done by hand, one ton of rock cut required 0.7 lb. of gelatine.

These figures show that with machine drilling considerably more explosive is used per ton of rock; this is because the cut is almost burned out, whereas with hand drilling the bottom of the shaft is advanced in benches, so that the rock is practically broken by stoping.

The highest rates of sinking in vertical shafts which have yet been attained by machine drilling, viz. 135 feet during September 1896 in the Durban-Roodepoort Deep, and 141 feet during December 1896 in the Vogelstruis Deep, were got by drilling the face over in one shift and blasting with heavy charges of explosives.

No. 1 shaft of the Jumpers Deep was sunk from surface to a depth of 511 feet, and No. 2 shaft to a depth of 461 feet, a total depth in the two shafts of 972 feet, by hand labour, at a cost of £20 : 18 : 8 per foot, including general costs and cost of pump chambers; the next 1345 feet, sunk in the two shafts up to 30th September 1896, were done almost entirely with machines, at a cost of £26 : 19 : 3 per foot. The average rate sunk by hand was 65 feet per month, and that by machines, in No. 2 shaft for 799 feet sunk, was 66 feet per month.

In the West Roodepoort Deep, during February 1897, No. 1 shaft was sunk 99 feet by hand at a cost of £12 : 5 : 3 per foot, and No. 2 shaft 88 feet by machine at a cost of £16 : 1 : 10 per foot.

In the matter of cost per foot, shaft-sinking by hand labour in the ordinary quartzite comes out considerably cheaper than by machine drilling. Sinking, however, through hard and compact igneous rock is done cheaper and quicker by machines, but such rocks have not often been met with; their occurrence and effect in the shafts of the Glen Deep and Vogelstruis Deep properties were noticed on p. 192.

It is more difficult with machine drills to keep the shaft as small, or the sides as regular, as with hand drilling, so that more blocking and packing is required behind the timbers; this is more especially a consideration in incline shafts where the track has to be laid on one side of the shaft.

Blasting the heavy charges placed in machine-drilled holes causes more displacement and damage to the timber, though, with proper protection by iron plates, the extra cost so caused is inconsiderable, and in incline shafts, with practically no timbering, it is not worth taking into account.

The fastest vertical sinking on the Rand has been done by hand drilling,[1] viz. during the month of August 1897, when 164 feet were sunk in the West shaft of the Angelo Deep, making a total depth of 955 feet in that shaft.

[1] See footnote on p. 193.

CHAPTER IX

UNDERGROUND STATIONS AND ORE BINS

A "STATION" is generally understood to be an enlarged chamber, cut out where a level starts from a shaft, and used for a landing-place. It answers pretty nearly to the Cornishman's "platt." Where there is an ore bin under the landing-place such a station is called a "loading station."

Stations.—In order that any level may be open to all the compartments of a shaft, it is necessary that the station should be of a width equal to the full size of the shaft. Its length is, with an ordinary-sized shaft, about 20 feet, and it is made high enough to give sufficient head room for the easy handling of everything which has to be landed. The usual heights are about 8 feet for an incline shaft, and 10 feet or 11 feet for a vertical shaft, though this height will probably be exceeded if double or three-decked cages are to be used for the conveyance of men in the deeper levels.

In a vertical shaft the ordinary distance between sets, 6 feet, does not give sufficient height, so a wall plate is taken out and longer studdles are set up, into which the dividers are mortised; these studdles are braced across the shaft by bolts underneath the dividers and end pieces. The station is made on the same side of the shaft as is the reef at that level, and as the shafts are placed with their longer sides parallel to the formation, the connection with the main drive on the reef is made by continuing a cross-cut from the back of the station, in a line at right angles to the length

of the shaft, as shown in Fig. 68. This is almost invariably the plan on which the stations and approaches are laid out for vertical shafts.

For incline shafts the best station is made by driving a heading horizontally out into the roof, with a width which is generally the full width of the shaft, as shown in Fig. 69, but which in some cases does not include the pump and ladder-way. This heading is made not less than 20 feet long and about 7 or 8 feet high, greater height being unnecessary because the angle which the incline makes with the floor of the station gives plenty of room for handling anything from the skips.

At less important stations, and where levels start off on either side directly from the shaft, the mouths of the levels are used as landing-places, as in Fig. 70. It sometimes is planned, as in the East Incline of the Robinson, that ore is hoisted some distance up one incline, and then tipped to be trammed to another shaft for further hoisting; in these cases a station is made in the footwall of the incline.

It is only where, as in the Princess Estate, Jubilee and Worcester mines, there are no bins underground, that the ore is hoisted or loaded from the station floors. In these cases the floor is covered with iron plates, on which frogs are fixed to guide the trucks.

Where the ore is tipped directly from the trucks into the skip on the incline, it is sometimes arranged that the incline track can be broken and the back wheels of the skip let down until the skip stands at a good angle for easy filling. In the mines where this arrangement is in use, the Jubilee, Worcester, etc., the skips pass from the incline to the surface through vertical shafts, so that the back of the skip could not be cut away, this being the ordinary method of facilitating the filling of the skips on the incline. The following sketch (Fig. 71) shows the arrangement of such a station at the Jubilee.

The station is covered with iron sheets and the trucks

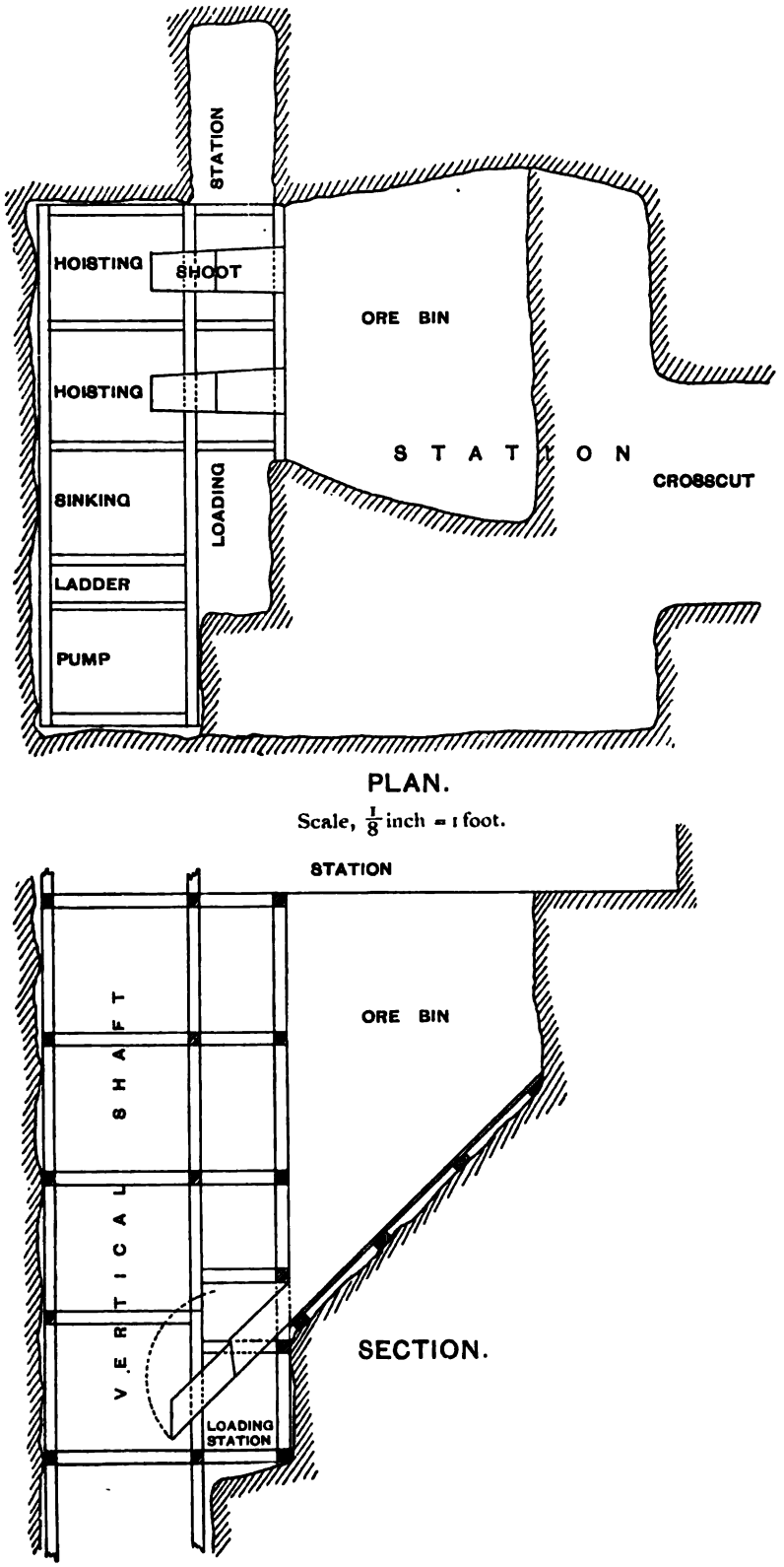


FIG. 68.—Plan and elevation of ore bin for vertical shaft.

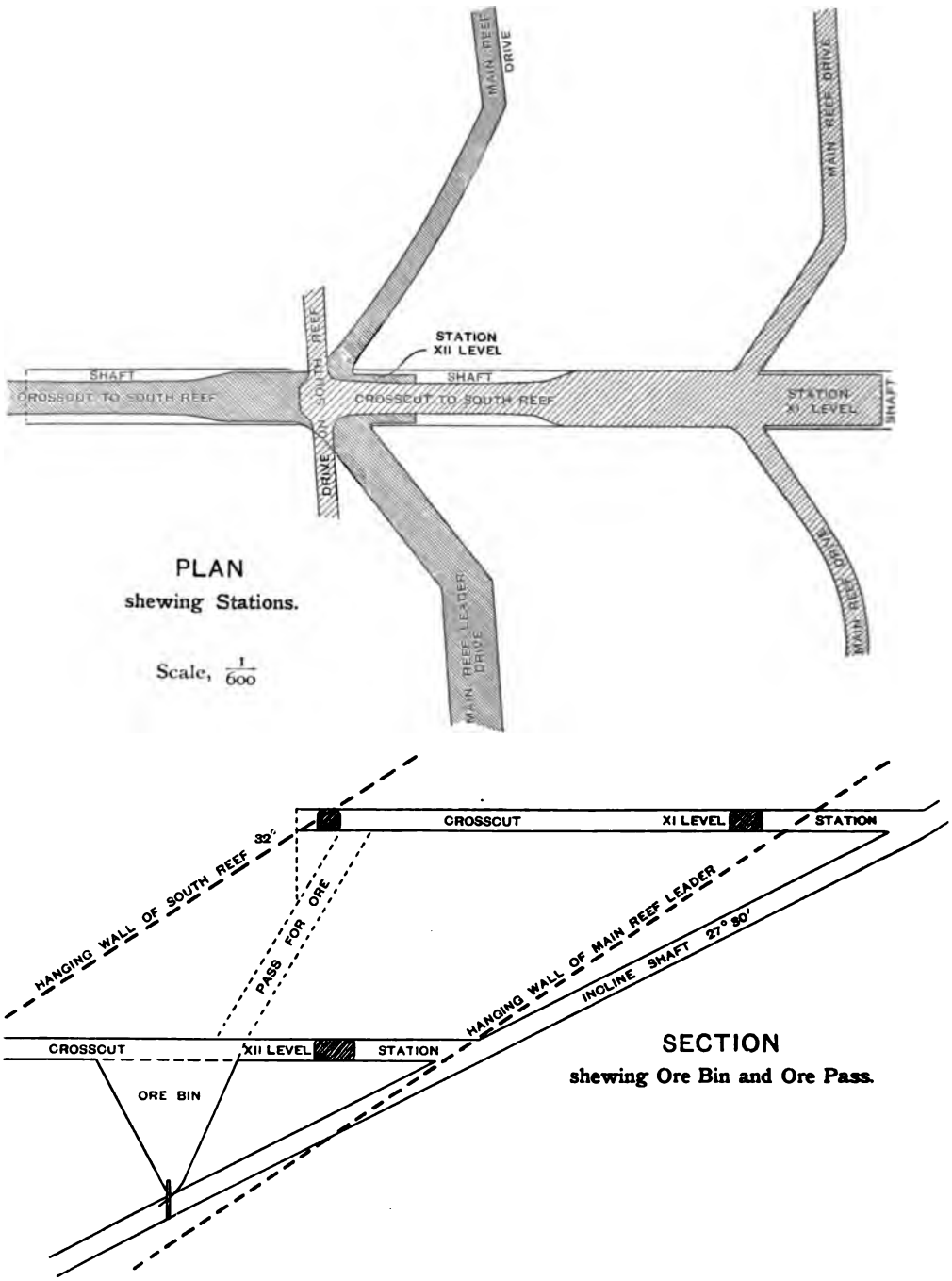


FIG. 69.—Stations, ore bin and pass in the West Incline shaft of the Robinson.

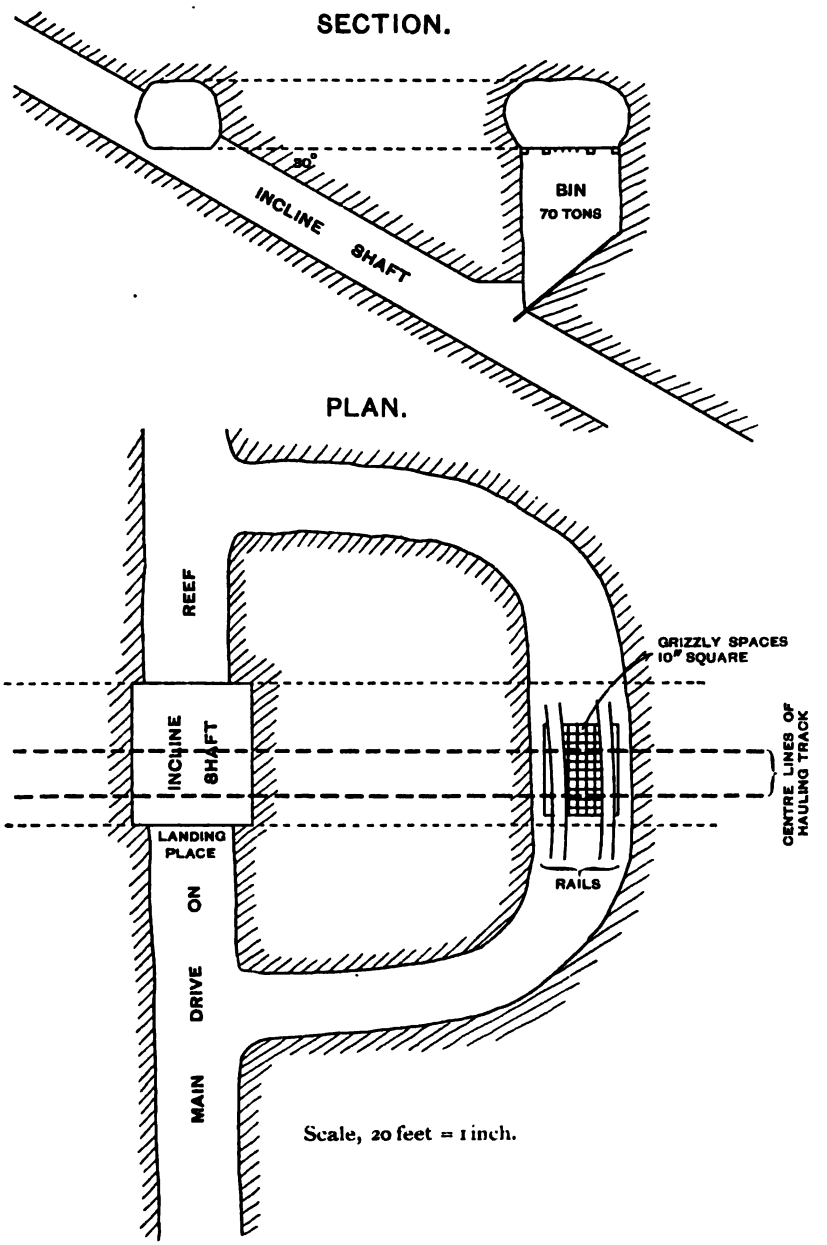


FIG. 70.—Plan and section of an ore bin over an incline shaft.

are brought alongside and tipped, end on, directly into the skip. In order to prevent any of the ore going down the shaft, an iron flap, with a rectangular space cut out so as to embrace the top and sides of the skip, is hinged to the station floor and turned down so that it rests on the front wheels of the skip; when not in use it is turned back so as to lie on the station floor.

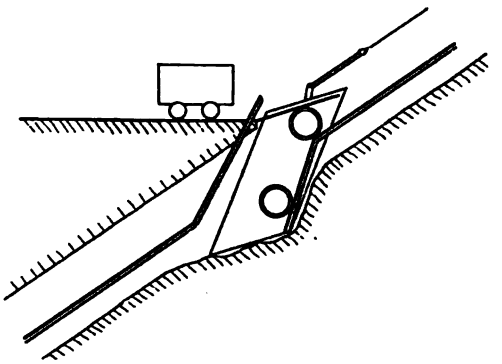


FIG. 71.—Diagrammatic section of a station on the incline shaft of the Jubilee.

In the Worcester, where side-tipping trucks are used, the ore is guided into the skip by a shoot, the weight of which is so balanced that when not in use it can easily be turned up out of the way of the shaft and as easily turned

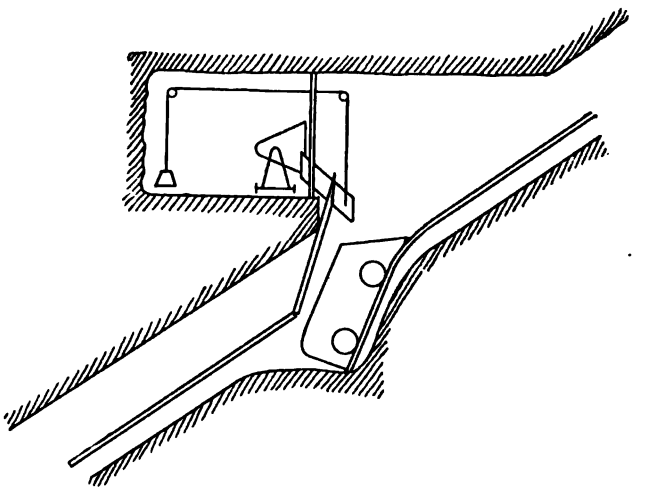


FIG. 72.—Diagrammatic section of a station on the incline shaft of the Worcester.

down when required. Fig. 72 is a sketch of a station in this mine.

This practice of breaking the track is not good where

large amounts have to be hoisted, because it takes away from the solidity of the track.

It is now the invariable practice in the larger mines to load the skips from ore bins.

Ore Bins.—Ore bins are made either by sinking a

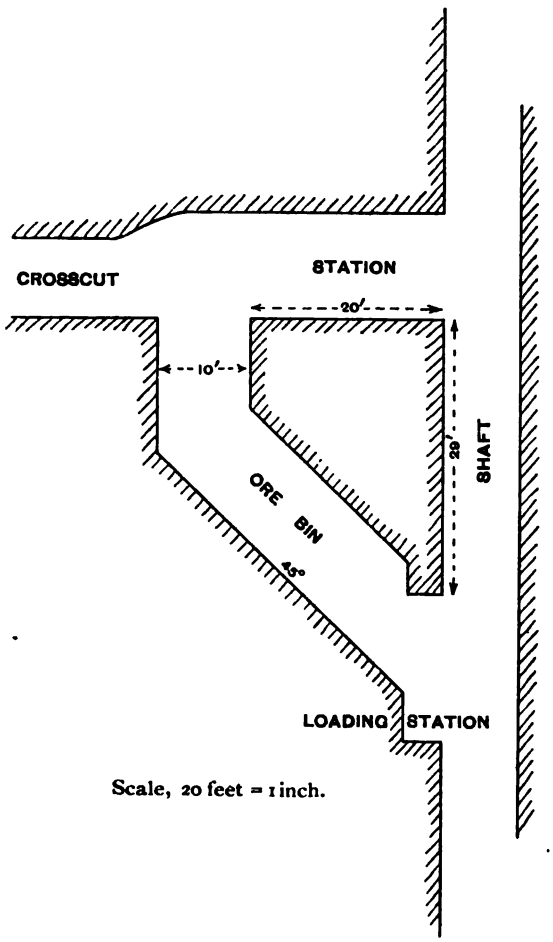


FIG. 73.—Diagram of a station in No. 1 shaft, Robinson Deep.

chamber or winze from the back of the station so as to break or open into the shaft, or by cutting away the angle under the station floor; in either case the bins break into the shaft at those compartments which are intended for hoisting purposes, as shown in Figs. 68 and 78.

Fig. 68 shows two views of a bin made for a vertical shaft by cutting away the angle. In this case the face of the bin is of timber, the main pieces of which are supported by distance pieces from the shaft timbering, and further secured in hitches at either end.

The rock is so cut that the bottom inclines towards the shaft at an angle of about 45°; it is usual to cover this with a flooring of stiff boards covered with sheet iron.

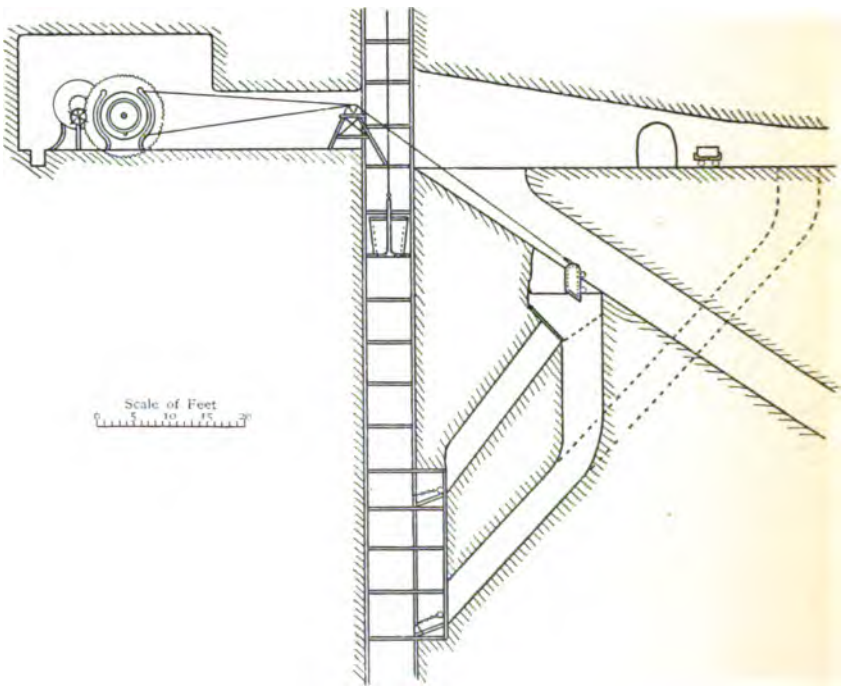


FIG. 74.—Section showing the main ore bins at the junction of the vertical portion of No. 2 shaft, Robinson Deep, with the incline portion, along which separate hoisting has been adopted.

The doors are manipulated from a platform which is as wide as the distance pieces, and which is extended on either side some distance beyond the hoisting compartments. Fig. 73 is a diagrammatic section of a station and bin in No. 1 shaft, Robinson Deep; the bin is here practically a winze from the back of the station to the shaft. This method leaves the floor of the station solid. It is further illustrated in

Fig. 74 (kindly made for me by Mr. A. E. Pettit), which is a section to scale of the main bins at the junction of the vertical portion of No. 2 shaft, Robinson Deep, with the inclined portion, along which separate hoisting has been adopted.

For incline shafts, after the bin has been cut, there are two main systems of arranging the face from which the ore is to be discharged; it is either made looking up the shaft, as in Figs. 70 and 75, or down the shaft, as in Figs. 76, 78, and 79.

The former method has up to the present been more

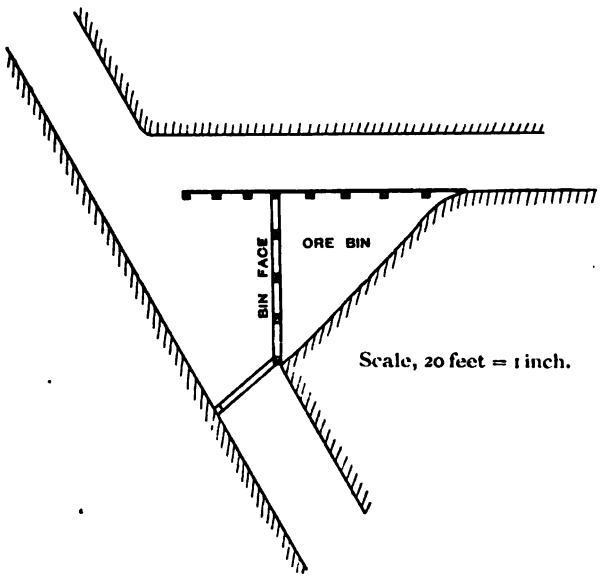


FIG. 75.—Diagrammatic section of a bin over an incline shaft dipping about 60°.

frequently used; it is an adaptation of a vertical bin to an incline shaft. The floor is inclined in the opposite direction to the shaft, so that the ore, as it falls from the shoot in the bin face, is deposited at the mouth of the skip, where its fall is broken. If the angle of the incline is sufficiently steep, the ore, by gravity, reaches the bottom of the skip; but after the higher inclination near the outcrop has been passed and an angle near 30° has been reached, the skip can only be loaded either by cutting away its back plate, which at the

same time lessens its capacity, or by forcing the ore down with poles.

When the face looks down the shaft, the ore has a free

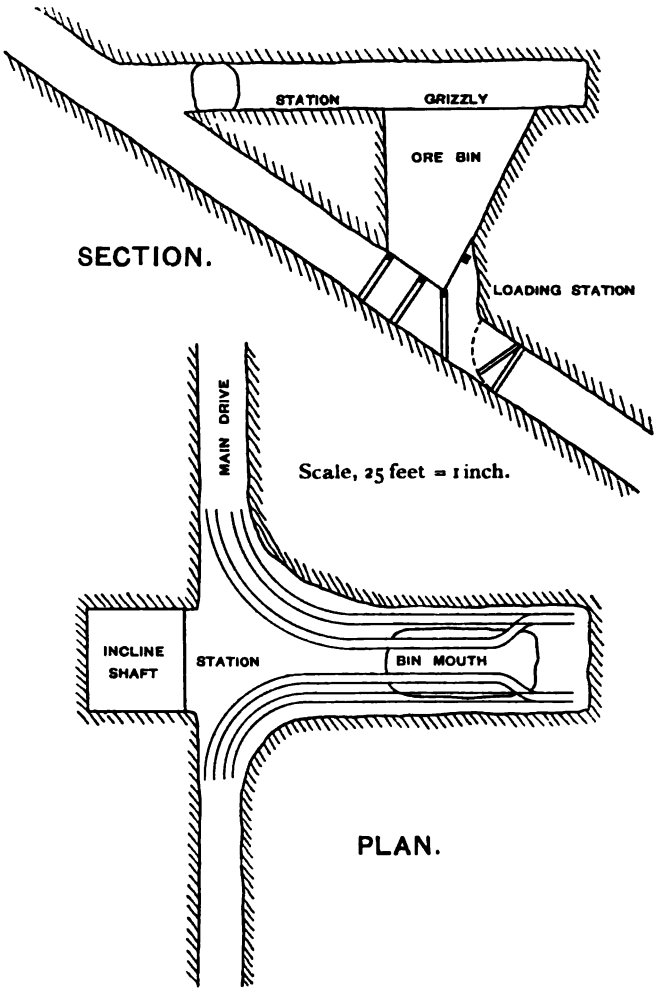


FIG. 76.—Plan and section of a station where the incline shaft is close to the reef.

fall right to the bottom of the skip; another advantage is that as the floor of the bin may be inclined with the shaft, a bin of greater capacity can be cut out in sinking from any one point on the station above, as seen from Fig. 67.

In a few instances, and only where the bins have been made by cutting away the angle of the station floor, the

discharge has been arranged through the floor of the bin, as indicated in Fig. 77, but this arrangement does not admit of an adequate regulation of the discharge, owing to the great weight on the door.

Fig. 78 shows a section through the bin made by sinking a winze from a station down to an incline shaft; the face, which looks down the shaft, is $9\frac{1}{2}$ feet across, and it has two doors. This bin has been made just at the junction between a vertical

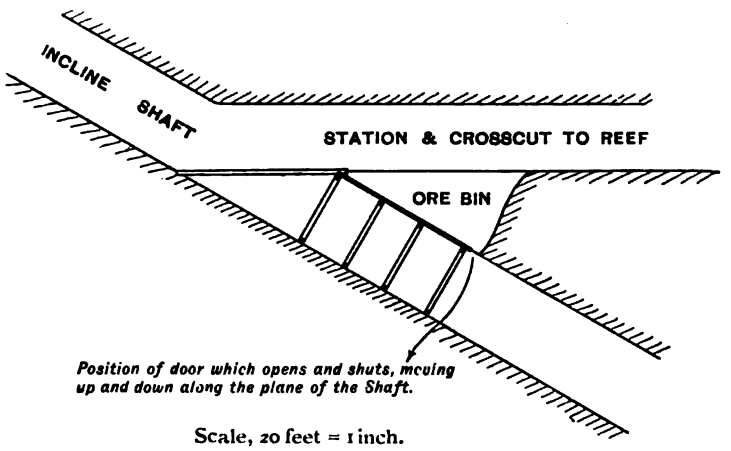
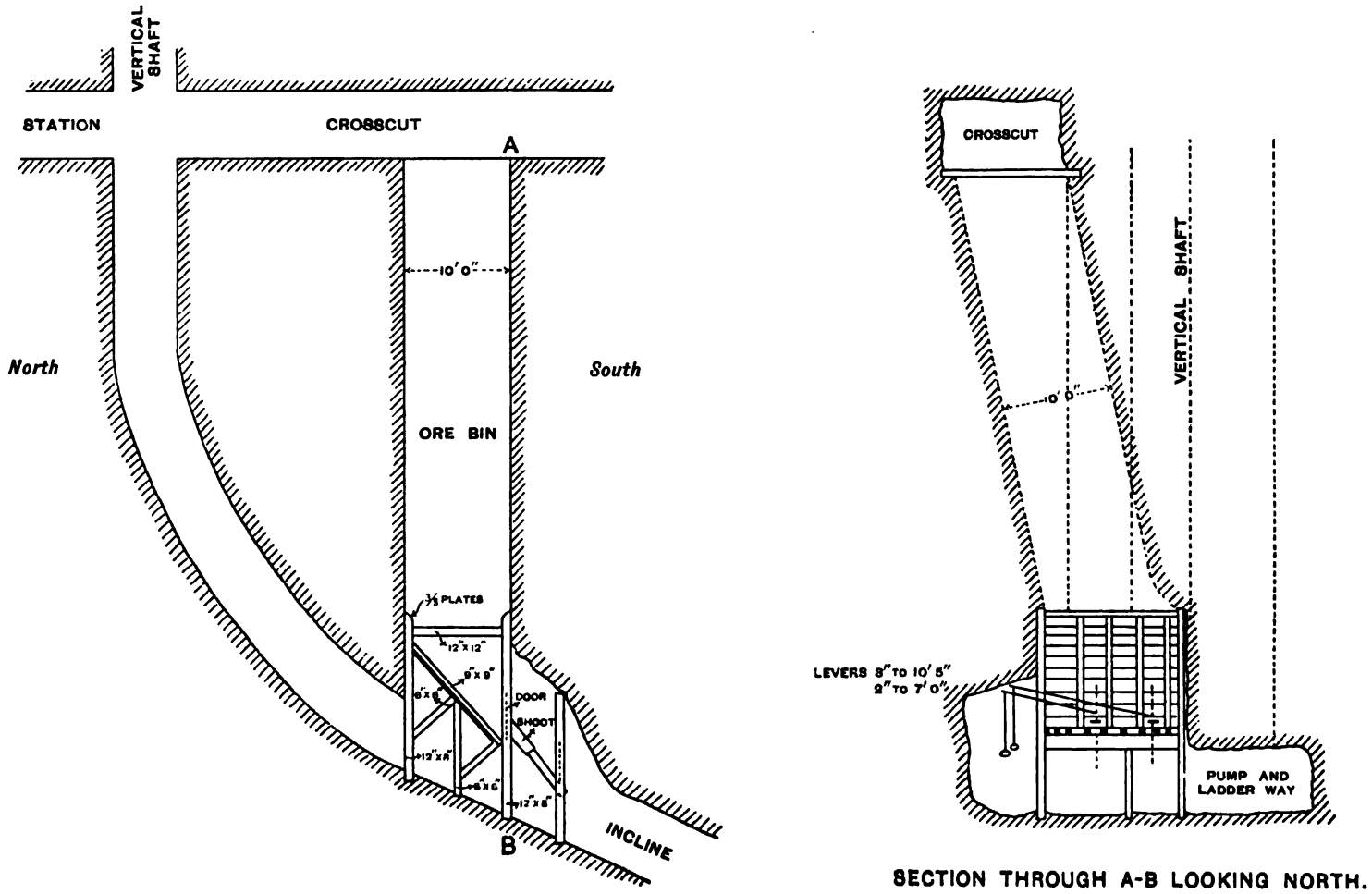


FIG. 77.—Diagrammatic section of an ore bin made over an incline shaft by cutting away the angle.

and an incline shaft. At other loading stations, bins similar to those shown in Figs. 76 and 79 are constructed. Figs. 75 and 77 are diagrammatic sections through bins which have been made by cutting away the angle between the station floor and the incline shaft. This method is better adapted for use in those shafts which are more steeply inclined, for at a lower inclination the angle cut away provides little capacity, and the bulk of the weight of the rock in the bins has to be supported on timbers.

The mouths of all bins have the necessary timbers across them to support the rails on which the trucks run to discharge the ore, and, in addition, they are crossed with heavy iron bars, generally stout rails, placed at such distances apart,



Scale, $\frac{1}{16}$ inch = 1 foot.

FIG. 78.—Arrangement of an ore bin over an incline shaft.

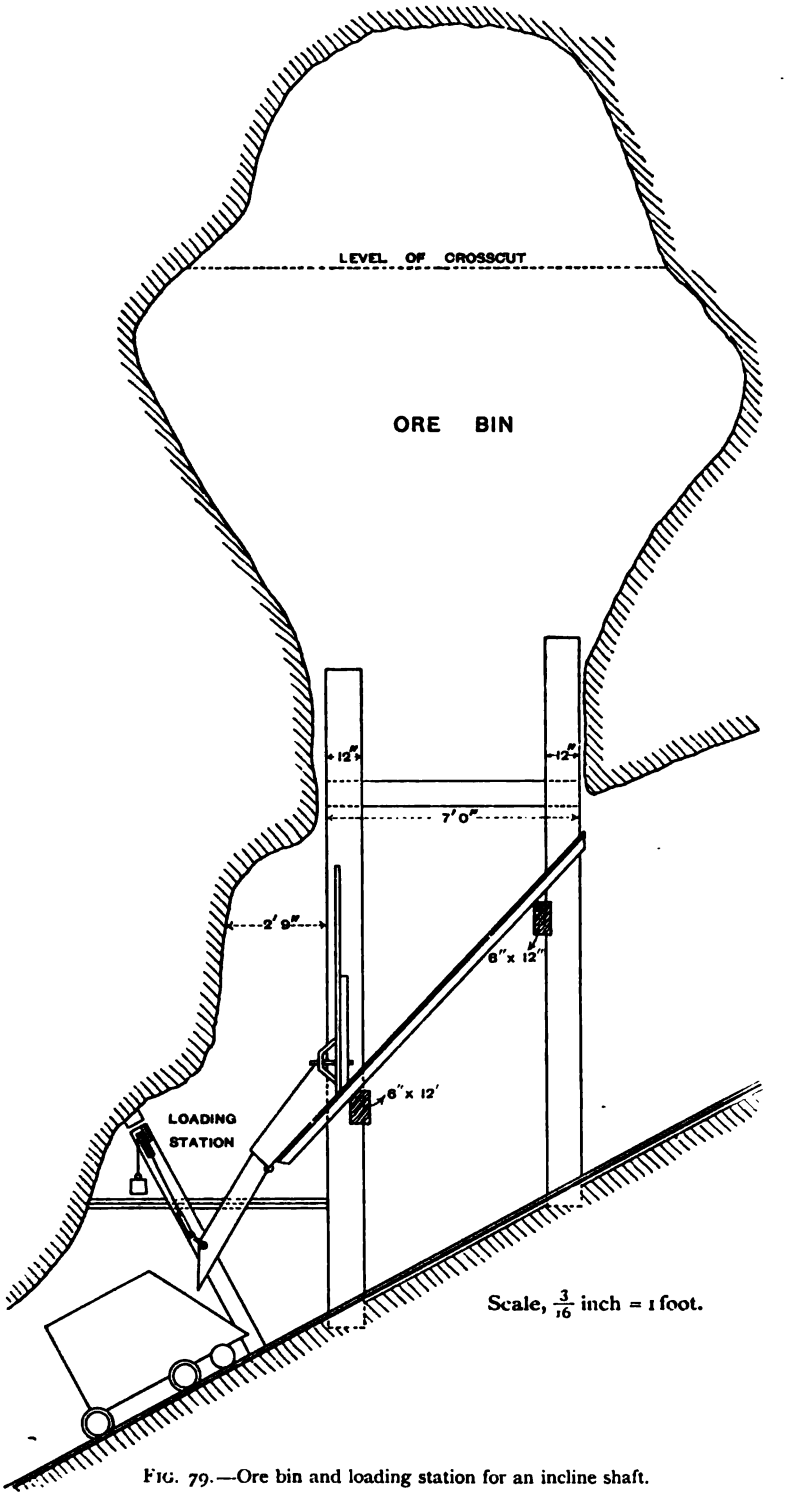


FIG. 79.—Ore bin and loading station for an incline shaft.

P

as indicated in Fig. 70, that all large lumps which would block

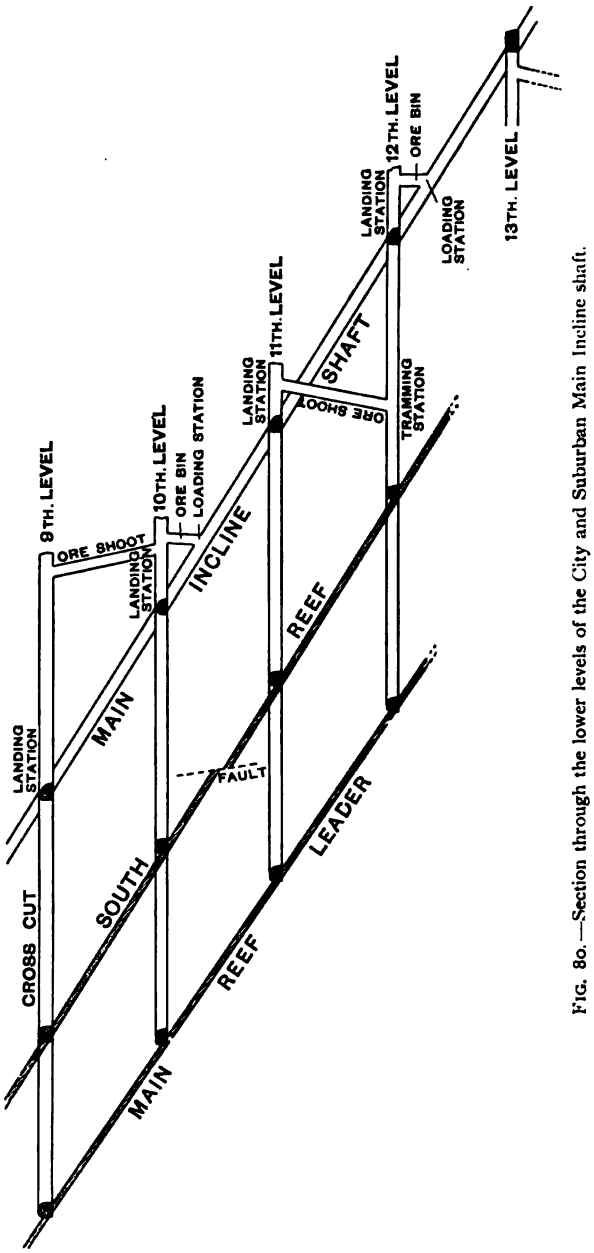


FIG. 80.—Section through the lower levels of the City and Suburban Main Incline shaft.

the shoot are separated, to be broken up before being passed into the bins.

In order to lessen the number of loading stations, it is now often arranged that the ore from one level is passed down to the bin on the level below; such a system of working the levels in pairs for hoisting purposes reduces the number of stations to one half, producing great simplification in all hoisting arrangements.

It is well illustrated in Fig. 80, which is a section through the lower levels of the City and Suburban Main Incline. Down to the 8th level, each level had an ore bin, but on the 9th level the main cross-cut was carried so far south, that the bin on the 10th level was reached by putting down a winze at an angle of about 80°. Into this bin the ore from the 9th level is passed, and thus the station at that level is done away with, and the bin capacity of the 10th level increased by the capacity of the winze. Similarly, the ore from the 11th level is passed to the 12th, with this difference, however, that instead of extending the cross-cut, the winze is sunk at a point near the shaft so that the ore, on its arrival at the 12th level, has to be trammed south to the bin. Apparently, of the two, this latter method is considered the better, for it is being repeated on the 13th and 14th levels.

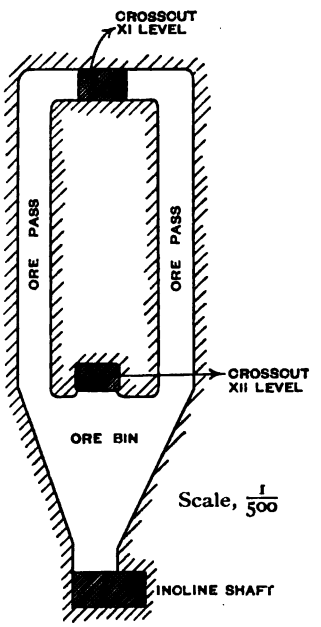


FIG. 81.—Projection on to a longitudinal plane of the ore passes and ore bins of the Robinson mine such as are shown in section by Fig. 69.

In the Robinson mine this plan has been adopted for use in the lower levels. There, on either side of the main cross-cut of the upper level (of each pair), a pass, inclined at a high angle, is made to connect with the ore bin at the lower level, as shown in Fig. 81. By dividing the bin below,

one pass can be used for development or waste rock and the other for ore.

Capacity of Bins.—The bins in the lower levels of the Robinson will hold 300 tons without including the capacity of the passes; those of the George Goch 200 tons. The

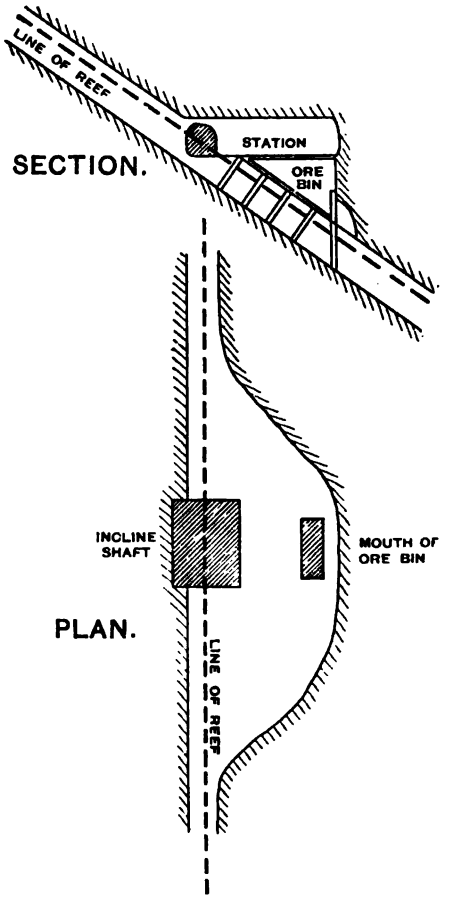


FIG. 82.—Diagram illustrating the connection between the ore bin and main drives when the shaft is on the reef.

average capacity of the Crown Reef bins on the incline is close on 200 tons, those now being made on the lower levels holding as much as 400 tons. In all the deep levels, bins of similar and even larger capacity are being constructed.

In the smaller mines bins of from 50 tons and upwards are in general use. In the Durban-Roodepoort the bins are

practically small shoots capable of holding, on each level, about 4 tons.

Where development work and stoping are proceeding along the same level, the bin is often divided into two, in order to keep the ore distinct from the waste rock.

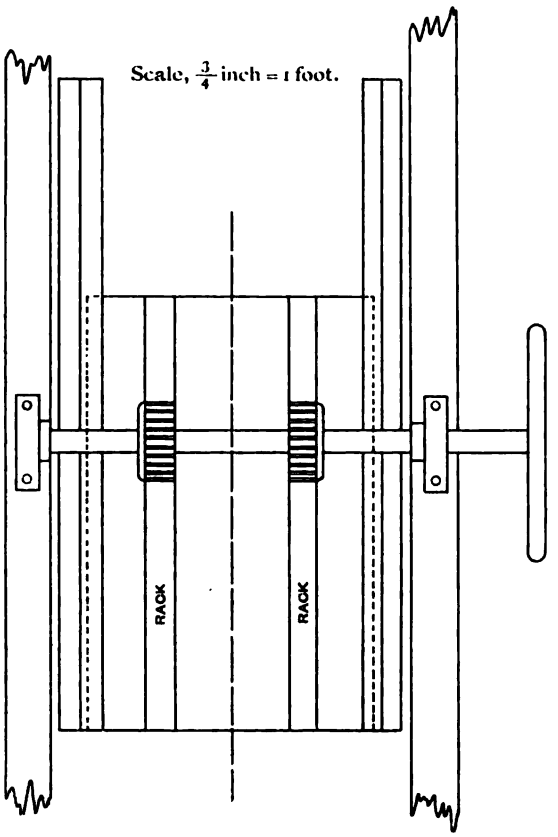


FIG. 83.—Bin or hopper door.

The connection between the main drives upon the reefs and the ore bin depends upon the position of the shaft relative to the reef.

With a vertical shaft the ore bin and station are, as stated before, made on that side of the shaft which is nearest to the reef, and the cross-cut is laid out along what is expected to be the direction of the shortest distance between the shaft and reef.

With an incline shaft the mouth of the ore bin is always in the roof of the shaft, whereas the reef may be either in the roof or in the floor; or with two reefs one may be in the roof and the other in the floor. In this last case connection is made with the ore bin and the two reefs, as in Fig. 69, which also illustrates each case separately. Fig. 70 also shows a method

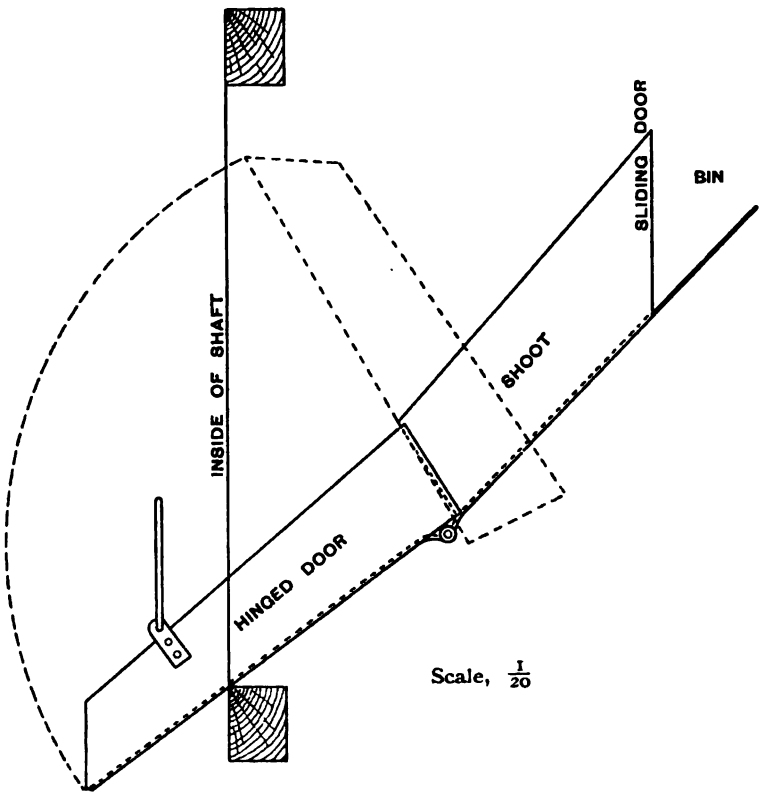


FIG. 84.—Arrangement for loading a skip in a vertical shaft.

of connecting ore bin, landing station, and main drives which is used in smaller mines, and Fig. 76 illustrates another method for larger mines.

Ore-Bin Doors and Shoots.—In the bin face sliding doors are placed. These are usually of $\frac{1}{2}$-inch steel plate, 2 feet wide to suit the size of the skip, and about $2\frac{1}{2}$ feet long. They are generally moved by a simple lever attached by a link

to a pin on the plate, as shown in Fig. 85, and more rarely by a rack and pinion, as shown in Fig. 83. Leading from this door in the direction of the skip mouth, a shoot is fixed at about the same inclination, 45°, as the bin floor. In vertical shafts another shoot is hinged to the end of this fixed shoot, as in Fig. 84. This is balanced by a weight so that it can be easily turned up, to be out of the shaft, or turned down so

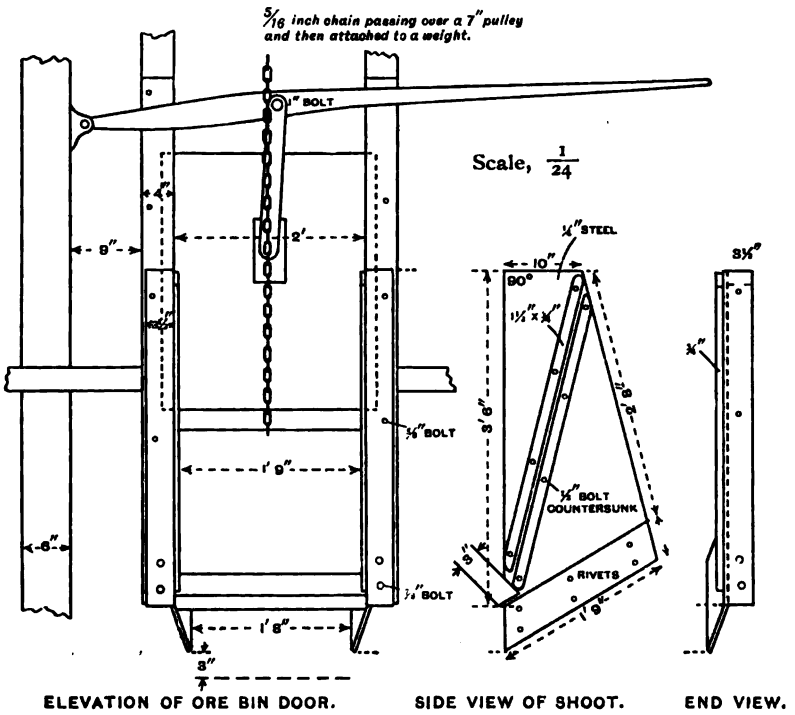


FIG. 85.—Underground bin doors at the City and Suburban mine.

that its lip is quite close to, and over the skip. Further, by manipulating this hinged shoot as a door to the fixed one, the fall of the ore from the bin door to the skip can be very well regulated. In incline shafts where the bin face looks down towards the bottom, these shoots are also used, as shown in Figs. 78 and 79. Sometimes, however, the fixed shoot is made longer, and the passage of the ore along it is regulated by additional doors called "spillage" doors. Thus in the City

and Suburban the ore from the bin passes through three doors before the skip is reached, viz. one bin door, one small shoot door, and one spillage door. In the same mine, where the bin face looks up the shaft, the spillage door is not used. The arrangement of the other two doors is shown in Fig. 85. Where a bin face looks up the shaft there is no need for the hinged shoot, for the ore has not a long drop, the fixed shoot being arranged so that between it and the skip there is no more clearance than is necessary.

Fig. 86 shows an arrangement of the timbering, bin doors, and loading shoots of a loading station; it is an enlargement of the upper portion of Fig. 67.

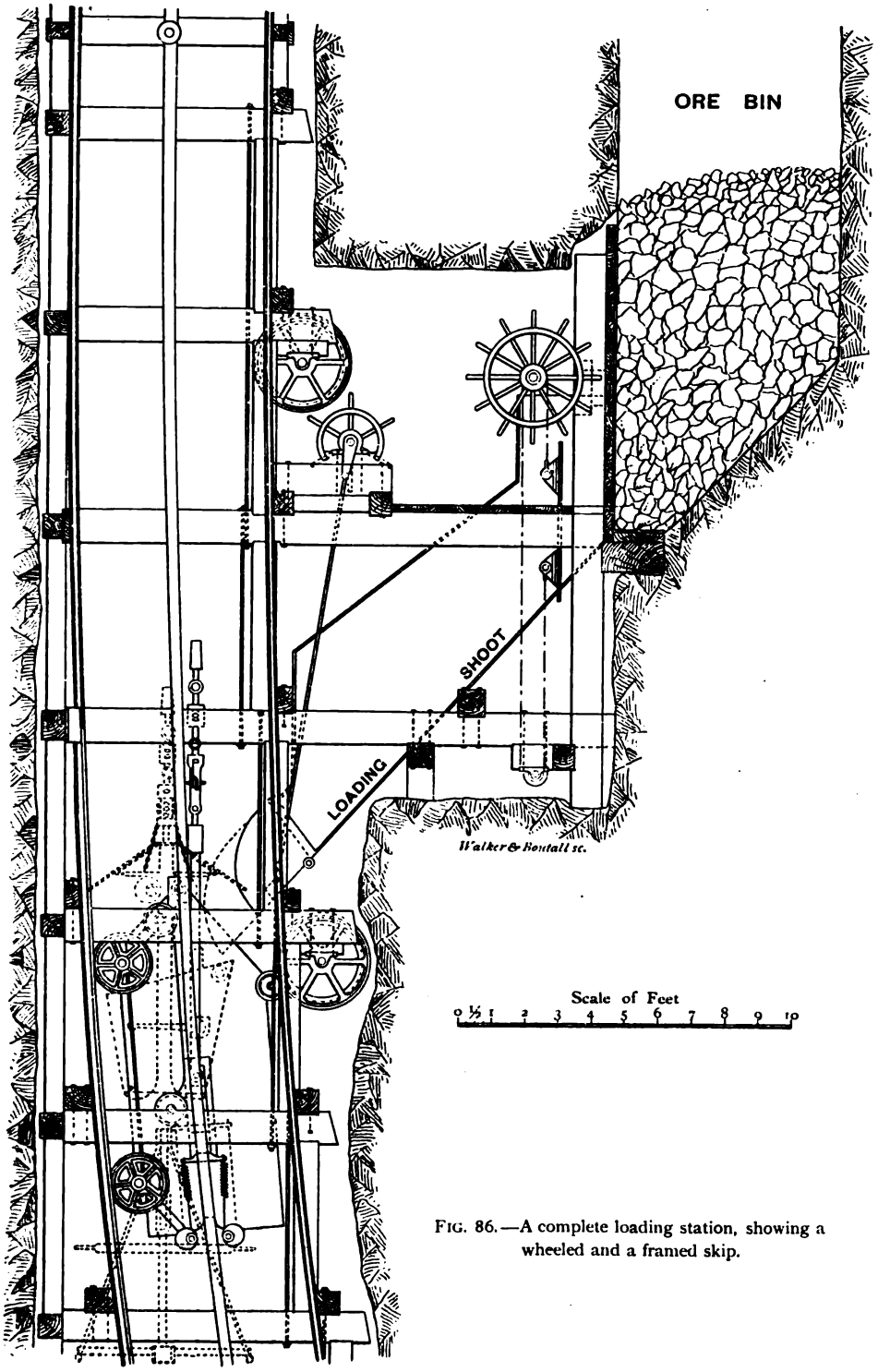


FIG. 86.—A complete loading station, showing a wheeled and a framed skip.

CHAPTER X

SHAFT TOPS AND HEADGEARS

Headgears.—A headgear is a structure erected at the mouth of a shaft to carry the hoisting rope and to guide the skip or cage, so that the ore and waste may be landed at such a height above surface as shall be convenient for its disposal.

The simplest form of headgear is that which is in use with a shaft which goes down from surface at a gentle incline, and where the ore and waste are hauled in trucks,

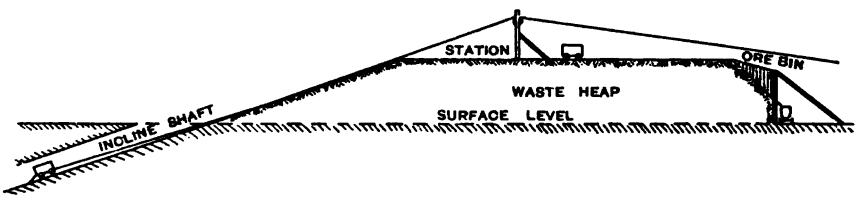
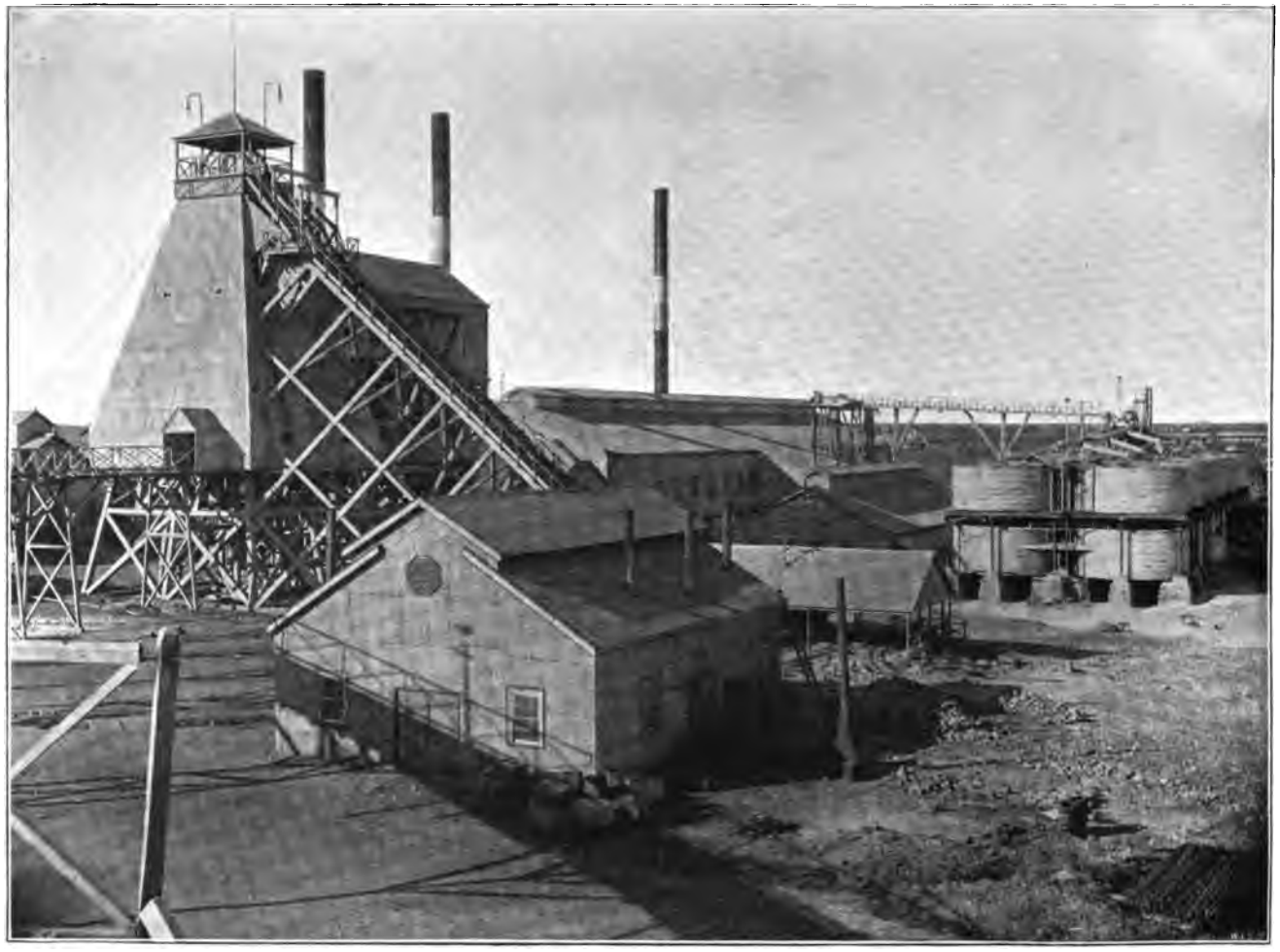


FIG. 87.—Diagrammatic representation of the method of receiving the ore at surface, as formerly used at the Nigel mine.

as at the Princess Estate and, until quite recently, at the Nigel mine.

Fig. 87 is a diagrammatic section illustrating the method which was used at the Nigel mine. The headgear there consisted of a simple upright framework carrying a pulley. This was erected on the top of a waste heap, at a level about 20 feet above surface, on which a small station covered with iron sheets was also constructed. The ore was hauled in a truck which was landed on to the station and then run off, so that its contents might be dumped into bins which were formed on



General View of the Surface Works of the Witwatersrand Mine, showing Headgear with Sorting and Crushing House attached, Mill and Cyanide Works.

the side of the waste dump. From these bins it was taken away in trucks running on the surface level.

Similarly the waste was hauled and tipped around the outside of the waste dumps. By this method, when the ore, after once having been at the top of the dump, again reached the level of the surface, it had been stored and the waste had been disposed of. Further advantage might have been taken of the height to which it had been raised to run it into an ore-dressing house, had such been in use. In the case of the Princess Estate the slope of the ground gives sufficient height for dumping without having to haul to the top of a waste-heap.

Such a headgear as this can, however, only be used where but little rock has to be raised.

Where larger amounts are hauled, and where the ore-dressing operations of screening, sorting, and crushing are carried out, it is now considered the best practice to raise the ore to such a height above the surface on the headgear that it may be tipped at once into bins, any further hoisting for subsequent operations being done separately. As the ore is tipped it falls on to grizzlies, through which the fines pass into bins, from which they are taken to the mill, whilst the coarse ore falls into other bins, from which it is taken to be dressed.

The height, which is sufficient for tipping the ore into bins, also allows the waste to be dumped, and leaves the actual mouth of the shaft clear for a landing-place for men and material.

There is an advantage in raising the ore at once to the limit of height required for all subsequent operations, in that such hoisting costs but little. As, however, the headgear has to bear all the strains and jars of rapid hoisting in the shaft, the higher the structure is, the very much stronger has it to be; and in order to take advantage of such a height once gained, the sorting and crushing floors must be quite close to

the headgear, or otherwise extensive trestle-work will be required. On mines where the ore is brought to the surface through one Main Incline shaft, as at the City and Suburban, Robinson, and Chimes West, the sorting floors and crushers can be conveniently arranged around the headgear inside of its main timbers; but where there are two or more main winding shafts a crushing station with sorting floors is best arranged at a central point between the shafts, in which case there would be no advantage in fully raising the ore on the headgear, for it would have to be brought down to surface level for transit to the crusher station.

Fig. 88 shows an incline headgear erected on the Driefontein, and Fig. 89 a vertical one erected on many of the Rand Mines properties. Both of these satisfy all the requirements of a headgear as indicated above.

This latter headgear is shown as it is arranged for sinking purposes when only country rock is being hauled. The bin is small, and from it the waste is trammed away at a convenient level.

Fig. 91 shows this same headgear arranged for dealing with waste and ore. Here, when ore is being drawn, the waste bin is covered over with a hinged shoot, over which the ore passes, to be divided by screening into three products, which after delivery into bins are trammed away along the surface level. When waste is being drawn, this shoot is turned back, and the contents of the skip are discharged into the waste bin, from which it is trammed to the dump, along a level which is at the same height above surface as that in Fig. 89. Sometimes for sinking, a temporary headgear, which is not capable of adequately receiving any quantity of ore, is erected, but where funds are sufficient it is considered the best practice to erect the permanent structure at once. Fig. 90 shows an early stage in the erection of one of the headgears on the Ferreira Deep which is to be permanent and complete at once. Fig. 93 shows the temporary wooden

headgear of No. 2 shaft, Robinson Deep, over which a large iron permanent headgear, 85 feet high, is in process of erection.

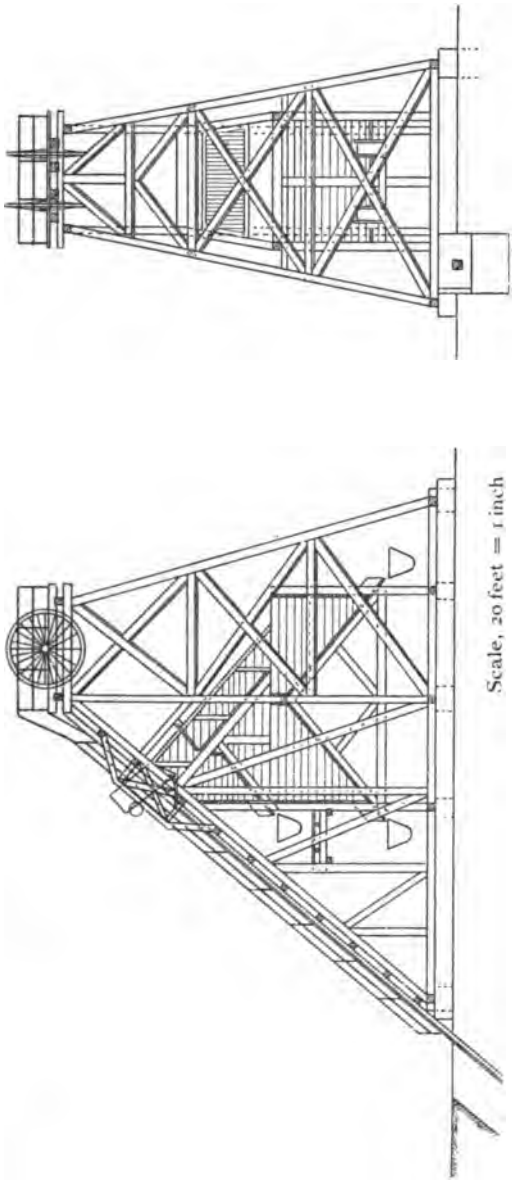
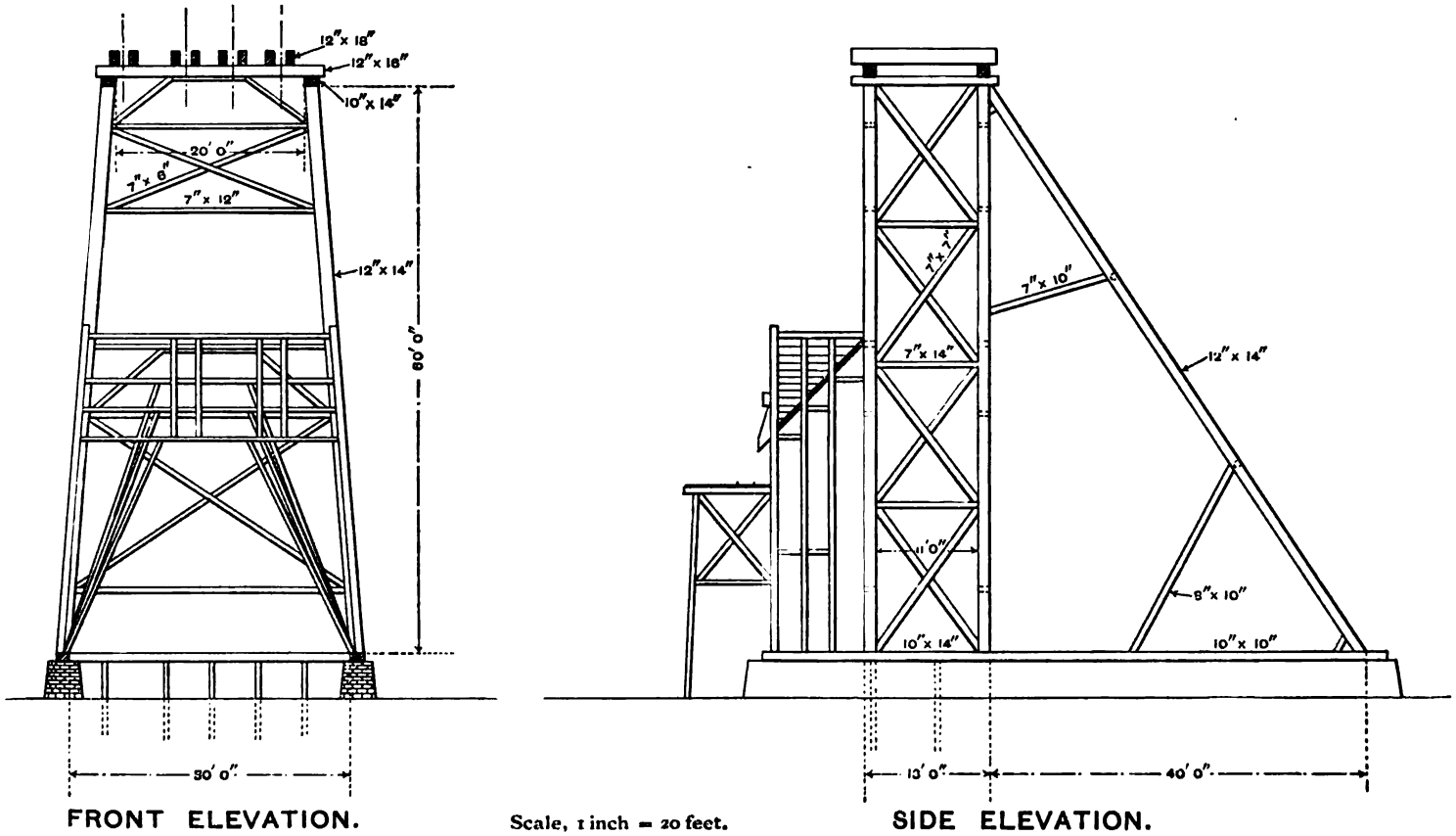


FIG. 88.—Inclined headgear, Drienfontein Consolidated.

In Fig. 92 are two drawings of a large vertical headgear, 85 feet high, which has been erected on the Knights Central,



FRONT ELEVATION. Scale, 1 inch = 20 feet. SIDE ELEVATION.

FIG. 89.—Two elevations of the style of headgear which has been erected on the Rand Mines properties.

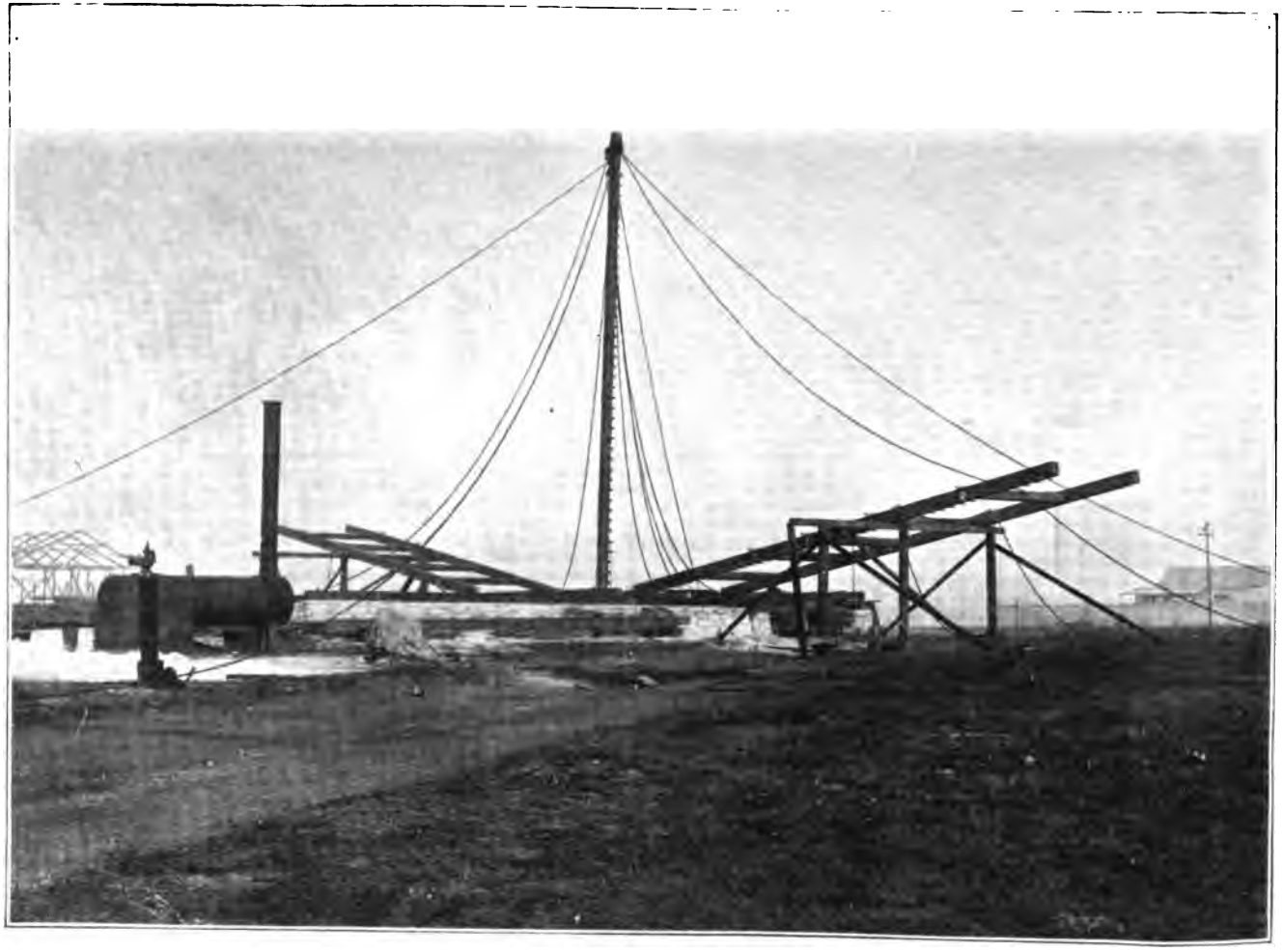
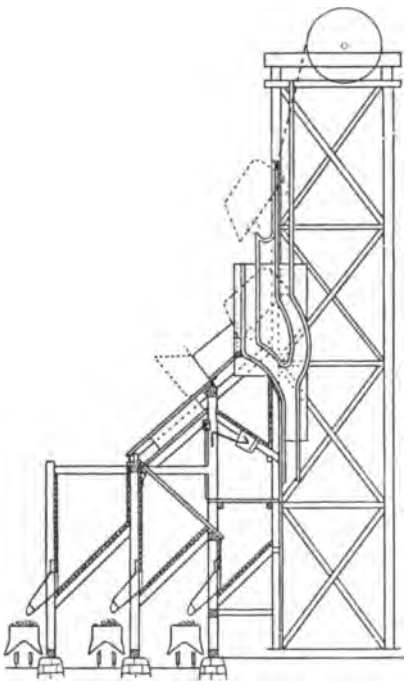


FIG. 90.—EARLY STAGE IN THE ERECTION OF ONE OF THE RAND MINES HEADGEARS, FERREIRA DEEP.

according to the designs of Messrs. N. Wilson and T. H. Leggett; a perspective view of this headgear is also shown in Fig. 96.

The advantage of having bin capacity in which to store the ore before sorting is shown by the practice in the Geldenhuis Estate, where the ore, drawn during both day and night from the mine, is stored in the bins at the headgear to be



Scale, 20 feet = 1 inch

FIG. 91.—Rand Mines headgear arranged with ore and waste bins and with tip for incline skips.

sorted during the daytime only, the result being a better product, obtained at less cost.

In some mines, especially those where the rock is hauled in trucks, there is little or no such bin capacity, but as this leaves the operation of sorting dependent upon a supply of ore which is generally irregular, this practice is not to be recommended, more especially where the area available for the display of the ore for picking is not great.

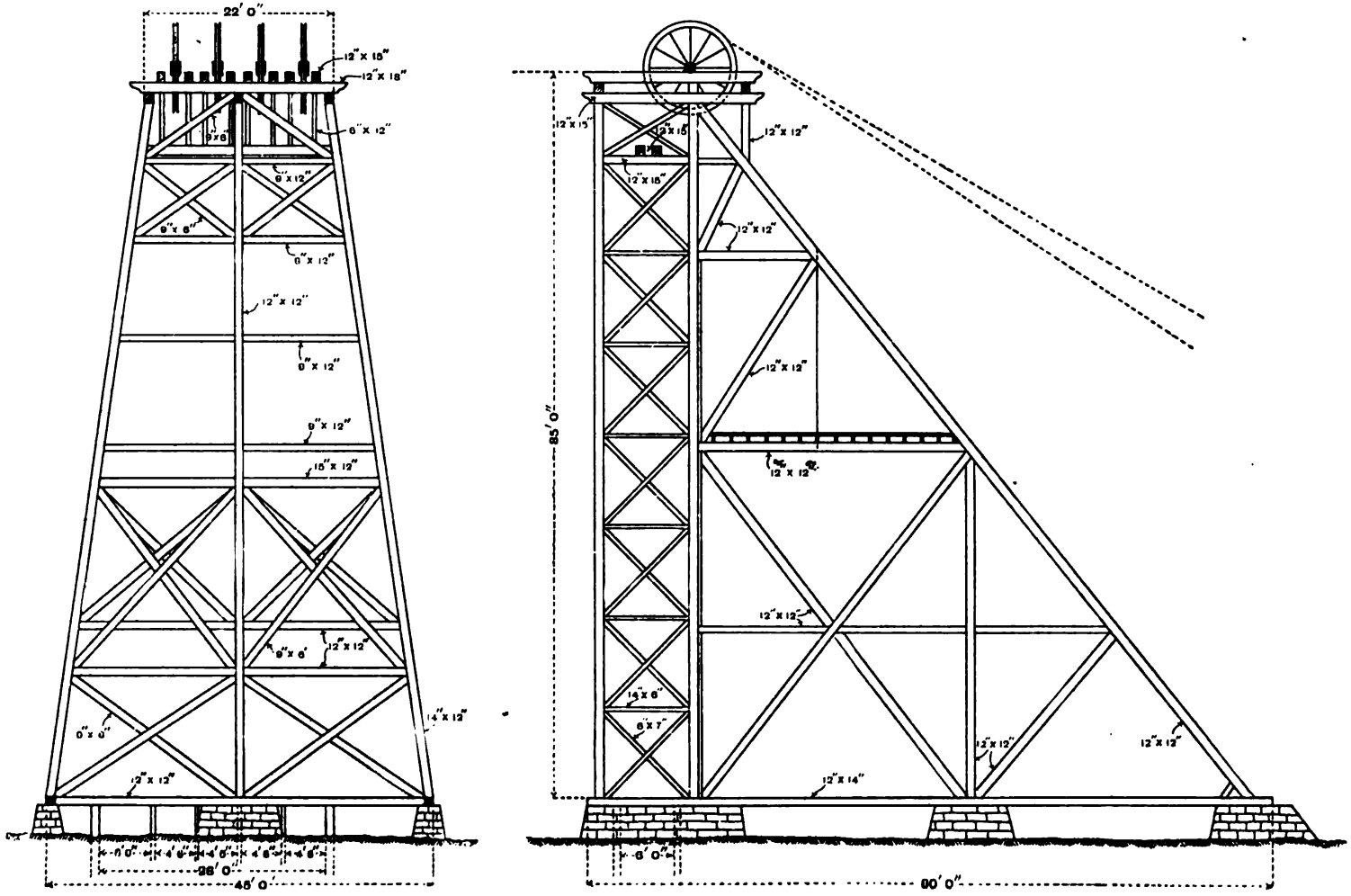
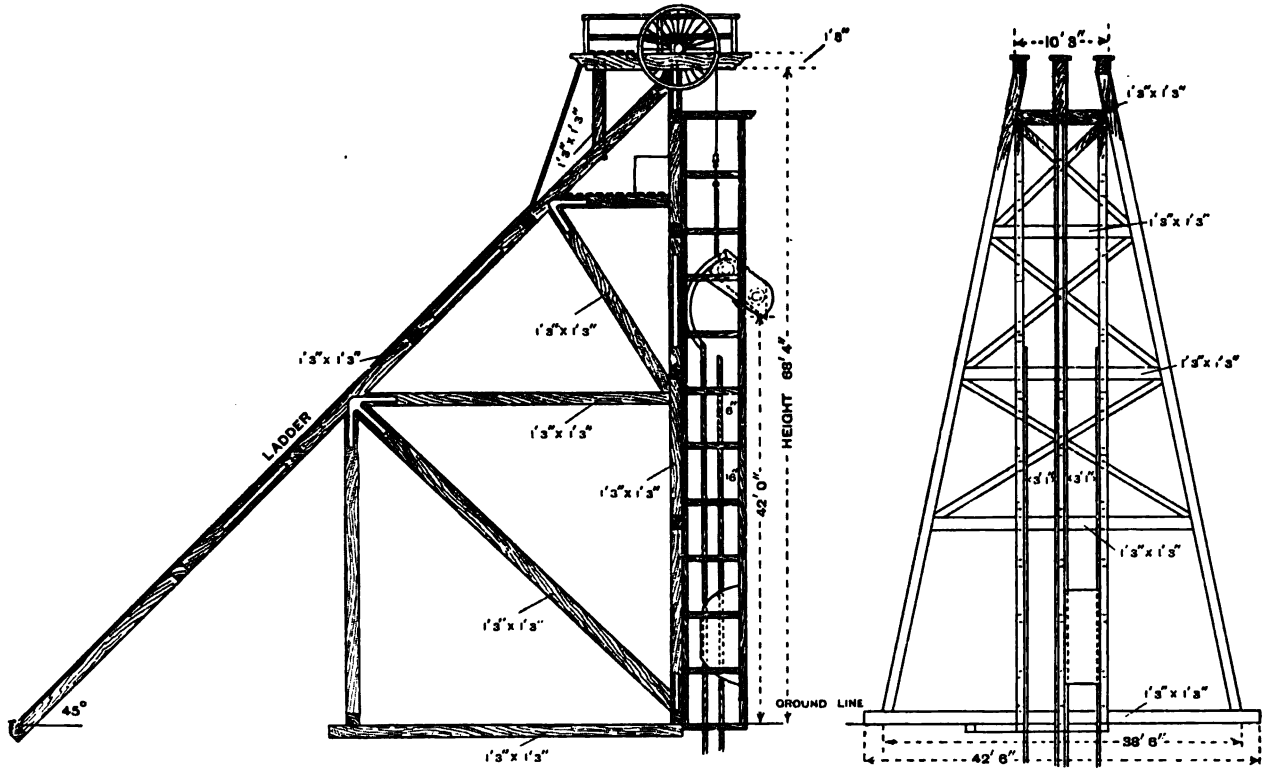


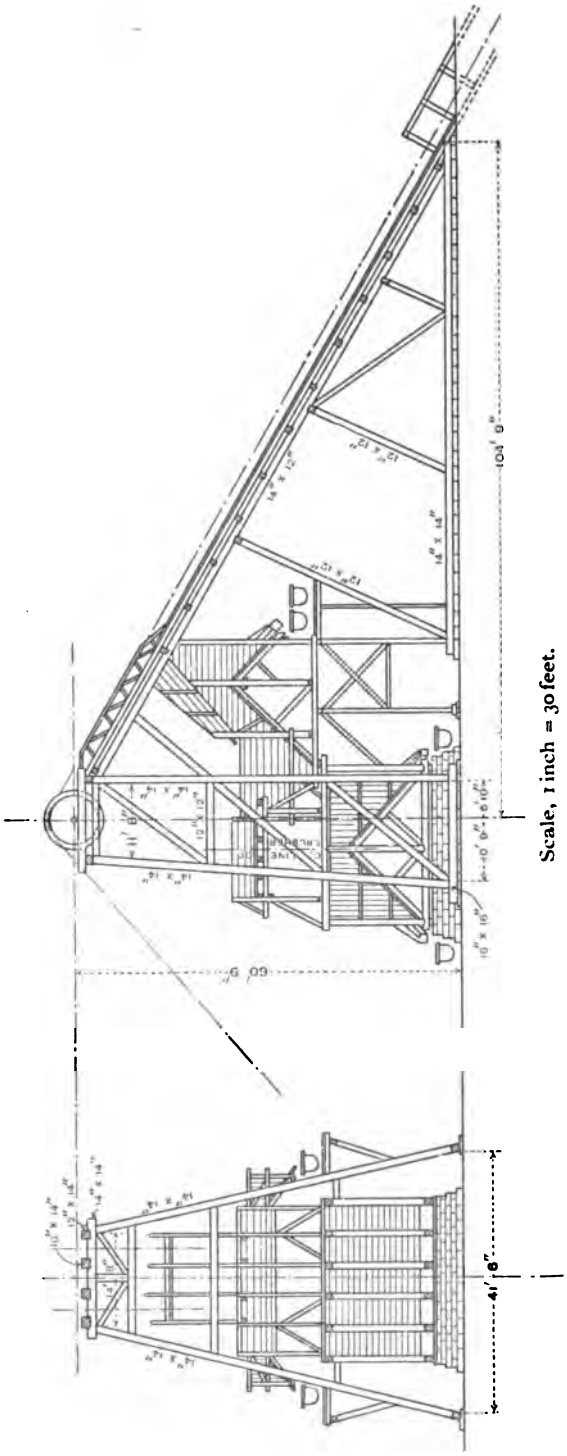
FIG. 7 Two elevations of a derrick erected on the Knights Central property.

FIG. 93.—STEEL HEADGEAR IN COURSE OF ERECTION ON NO. 2 SHAFT, ROBINSON DEEP, OVER THE TEMPORARY WOODEN HEADGEAR.



Scale, 20 feet = 1 inch.

FIG. 94.—Ferreira headgear.



Scale, 1 inch = 30 feet.

FIG. 95.—Two elevations of the City and Suburban Incline headgear.

The style of headgear in use at the Ferreira is shown in Fig. 94. At that mine, as the bin capacity above the sorting floor is limited, the ore is more regularly fed on to it by storing some of it in trucks when winding is proceeding rapidly, and by drawing on this store when but little ore is coming up.

At the City and Suburban, Robinson and Chimes West properties, as stated before, and at the Treasury, New Primrose, and other mines, the ore is hoisted to its full height at once, and when it again reaches the surface level,

FIG. 96.—VIEW OF THE HEADGEAR WHICH HAS BEEN ERECTED ON THE KNIGHTS CENTRAL MINE, TWO DRAWINGS OF WHICH ARE GIVEN IN FIG. 92.

it has been sorted, crushed, and stored, and is ready for the mill, the bins, crushers and sorting floor being on foundations independent of the headgear. In these properties most of the ore is drawn through one shaft, which makes this practice convenient, and there is no material bin capacity above the sorting floors, so that the height required is not excessive. In Fig. 95 two views of the City and Suburban headgear are shown, and the following is a description of that on the Chimes West, which was erected at the mouth of the Main Incline shaft from designs by Mr. B. H. Wright:—

A diagrammatic plan and elevation are shown in Fig. 97. Its chief dimensions are: vertical height from top of masonry to top of cross-pieces that support the pulley axles, 60½ feet; length over all, 105 feet; width of base under main posts, 36 feet; angle of hauling track to the horizontal, 40°.

Between the main timbers are the floors on which sorting and crushing are done, and bin capacity provided, all on foundations which are independent of the headgear.

Under all the mud sills there are masonry walls 2 feet wide on top and 2 feet high. The masonry of the crusher house is 7 feet 7½ inches above ground, and there are three walls under the mud sills 2½ feet wide at the top; this extra height of the masonry allows the trucks which take the ore from the bin to run along the level of the mud sills of the headgear timbers.

The four main posts are 12 inches by 12 inches in cross-section; the cross-pieces on top are 14 inches by 16 inches.

The skip tips the ore on to a waste door, 5 feet long, before coming to the grizzlies. This door is made of 9-inch by 1½-inch planks covered with ⅛-inch plate iron; when it is open the skip tips into the waste bin. The grizzlies are 11 feet wide at bottom, 13 feet at top, and 12 feet long, and they are set at an angle of 40°. The ore delivered on to them has not, as is usually the case, a free fall; it is boxed in and delivered through two shoots at the bottom into

trucks, which distribute it wherever desired along the sorting floor.

The sorting floor is at a level $5\frac{1}{2}$ feet below the bottom of the grizzlies; it has a clear space for sorting, which is 17 feet long and 27 feet across, with the exception of the mouths

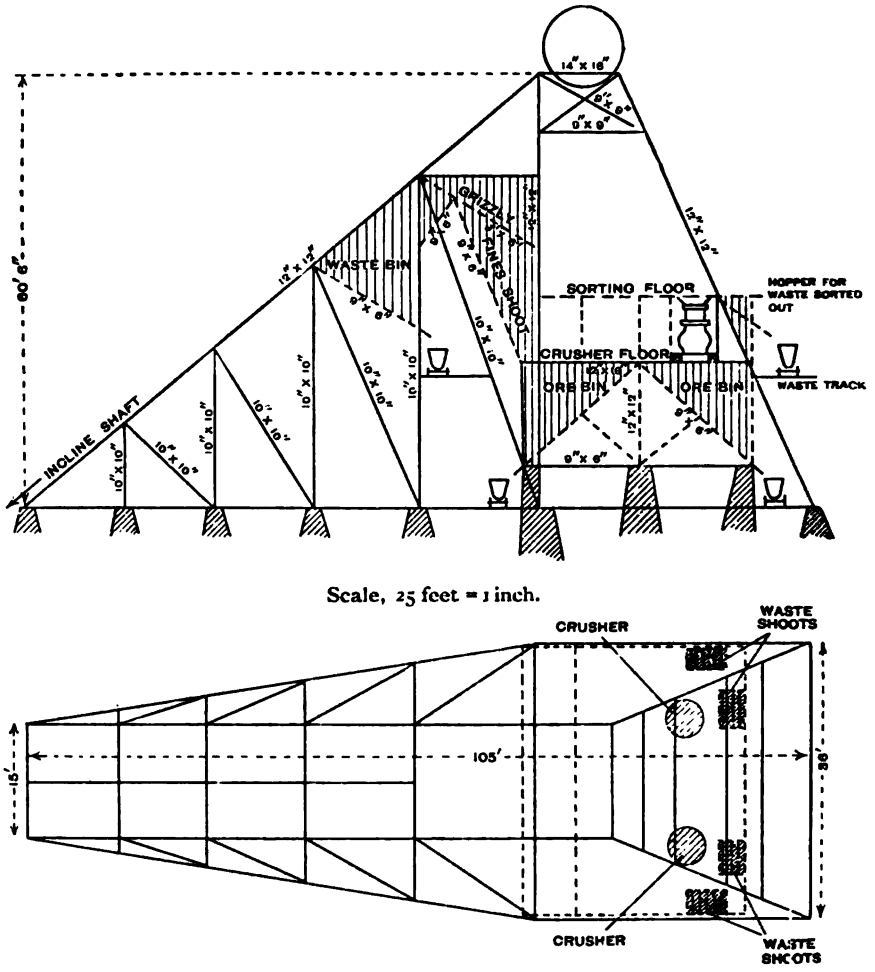


FIG. 97.—Diagrammatic plan and elevation of the Chimes West Incline headgear.

of the crushers, each 7 feet in diameter. Around the outside of this area there are four waste hoppers, including which the sorting floor is $22\frac{1}{2}$ feet long and 36 feet wide.

The ore bins are under the crusher floor. The ore which has been sorted and crushed falls directly into its bin, and the

fines pass down into their bin from the grizzlies through a shoot. These bins are shown in Fig. 108; they are lined with planks, 9 × 1½ inches, which are covered where the ore falls with ¼-inch iron plate, and in all other places with ⅛-inch plate, and the bottoms are set at an angle of about 38°. The

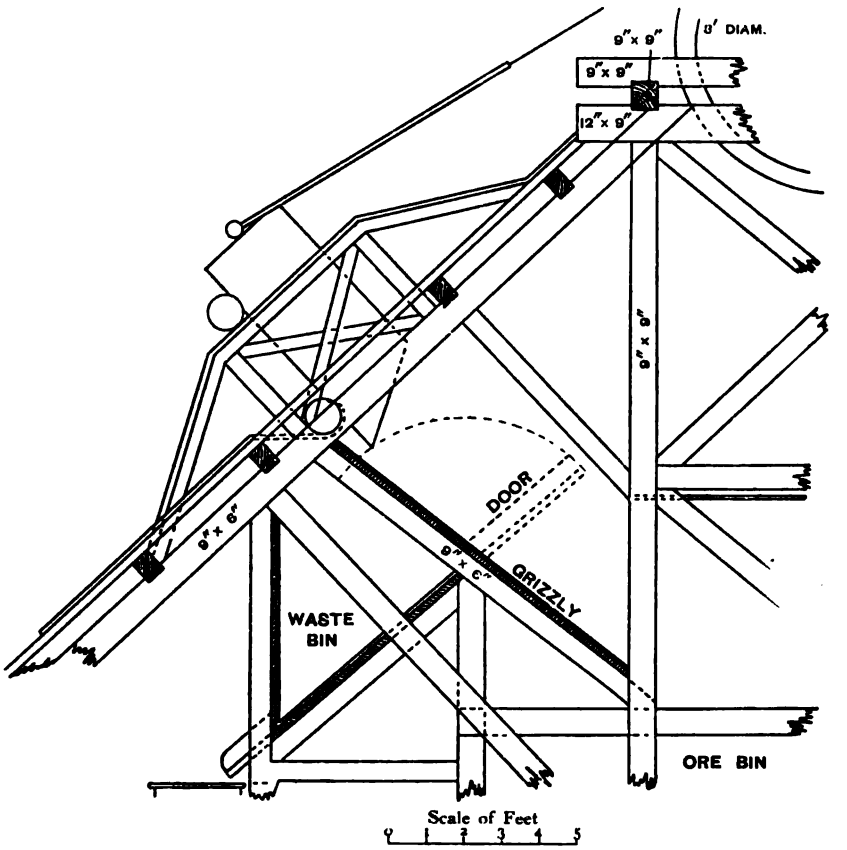


FIG. 98.—Tip on an incline headgear (Kleinfontein).

waste bin rests on the headgear framework; from it the waste is trammed away at a level 18 feet above the headgear masonry, and at this same level the waste which has been sorted out is also trammed away.

Tipping Arrangements.—The tip which is invariably used for hauling on the incline is shown in Fig. 98. The skip has four wheels; of these the two back ones

have a tread which is about twice the width of the front ones, as shown in Fig. 117. Arrived at the tipping point, the front wheels pass on to a short length of horizontal track, whilst the back wheels mount up along a track more steeply inclined, being prevented from passing on to the horizontal by the extra width of the tread, until the position depicted in the figure is reached, when the ore is tipped.

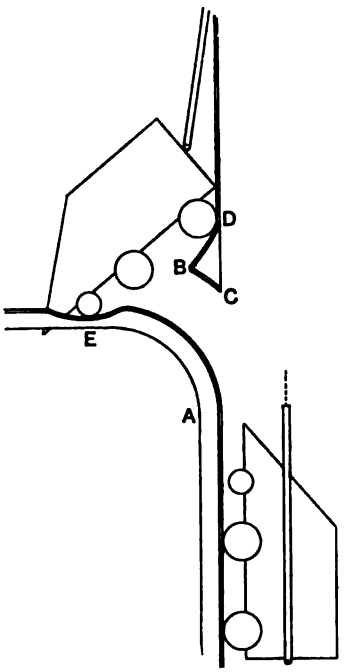
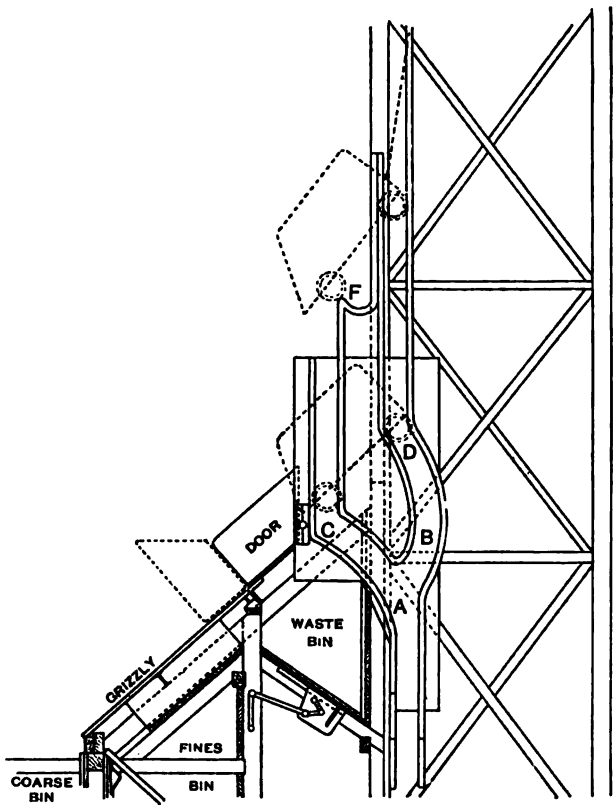


FIG. 99.—Arrangement on the head-gear of a vertical shaft for the tipping of a skip carried on wheels.

With vertical shafts the arrangement for the tip depends upon the kind of skip in use; one arrangement is shown in Fig. 99. In hoisting, when the front wheels have arrived at the point A, they turn out of the shaft to follow the curved track towards E, owing to the extra weight of the skip on that side, and to being guided in a manner indicated in Fig. 121, the nose of the skip at the same time running close to the BC. The back wheels, following, are lifted upwards so as to roll against the curved rail BC, the skip at that time being practically horizontal. Arrived at B, these wheels continue along BD, the nose of the skip at the same time getting lower till the front rollers are in the hollow at E, when the contents are completely tipped. On lowering, the skip falls automatically down again into its ordinary position in the shaft.

In Fig. 100 another arrangement is shown. This is used with a skip which is identical with incline skips. In hoisting, when the point A is reached, the front wheels roll along to C, the back ones at the same time being brought up to A. From

this position, in response to the direct pull, the back wheels are taken along BD, so that the skip turns around the front axle till a horizontal position is reached, when, in addition, the front wheels move up along vertical guides (the back wheels at the same time moving more rapidly) till a tipping angle



Scale, 10 feet = 1 inch.

FIG. 100.—Tipping arrangement on a vertical headgear for an ordinary incline skip.

is reached. Provision is also made for over-winding. When the front wheels get above their guides they are caught when coming back by a loop, as shown at F, and thus, should the rope be broken, the skip does not fall down the shaft.

When a skip such as is indicated in Fig. 101 is used, the tipping is arranged differently; the skip rests on the bottom of a frame, to which it is also hinged on one side as at A; when

in the vertical shaft it is secured by a latch which is pivoted on the frame, and which embraces at one end a pin, C, on the skip, and has at the other a roller, B.

When the skip approaches the tip, this roller is caught

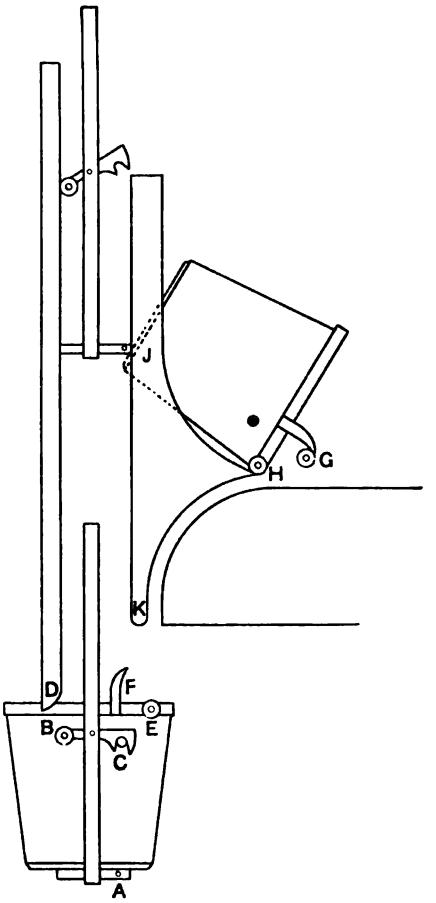


FIG. 101.—Diagram showing the method of tipping a skip which, in a vertical shaft, is carried on a framework.

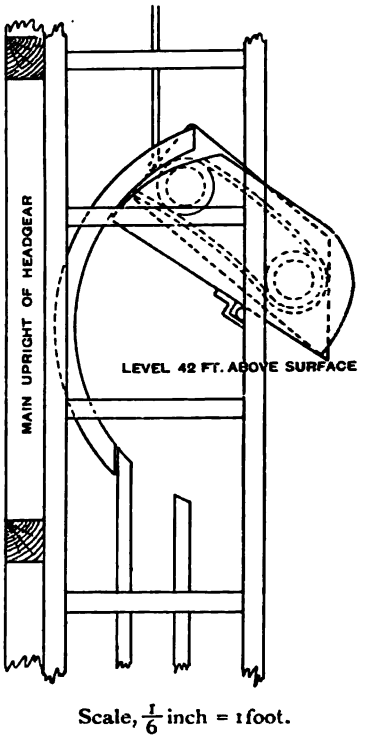


FIG. 102.—Arrangement for tipping the skip on the Ferreira headgears.

against the curved bottom of a guide, as at D, so that it is pressed down and the pin on the other side is released. There is another roller, E, on the skip, which, when the latter is about 2 feet farther up the shaft, enters and continues along a curved groove KH, so that the skip is caused to turn down about the hinge at the bottom, until it lies on its side. In

this position a horn, F, on either side, covers a roller, G, which, when the skip is further raised, it grips until the roller E is lifted, so that when the hold is released E falls easily on to the curved guide HJ, and the skip is tipped. On lowering, the skip automatically regains its position and the latch again secures it.

In the Ferreira, where skips on wheels are used, the tipping is effected by an entirely different arrangement, which is shown in Fig. 102. The wheels are guided up the shaft, till at the desired point on the headgear they are held on either side in a blind loop fixed to the sides of an iron envelope or tumbler, into which the skip has entered, and which is centred near its top, on the tipping side of the headgear framework. The independent motion of the wheels is thus passed on to this tumbler, which turns up until the contents of the skip are discharged. Should over-winding occur, this tumbler is further raised, so that the wheels escape again on the upper side; in this position the tumbler is secured, so that if the rope should break, the skip would fall down on to it, either to be held or to be turned off from the shaft. On the Jubilee headgear the contents of the skip are discharged through a hinged door, towards which the bottom of the skip slopes, at an angle of about 45°.

Where ore is brought to the surface in trucks, these are either constructed to tip easily, side-tipping trucks being the best, or they are tipped endwise by a tumbler such as is shown in Fig. 103, which is adapted for use with low rectangular trucks. There is the advantage with this latter method, in that less wear and tear is caused and less expensive trucks may be used. It is in use on the surface at the Princess Estate and underground at the Durban-Roodepoort, and will probably be used in the Robinson Deep.

In addition to the tip for ore and waste it is often advantageous to have one, generally at the surface level, for landing the drills. Fig. 104 shows an arrangement for breaking an

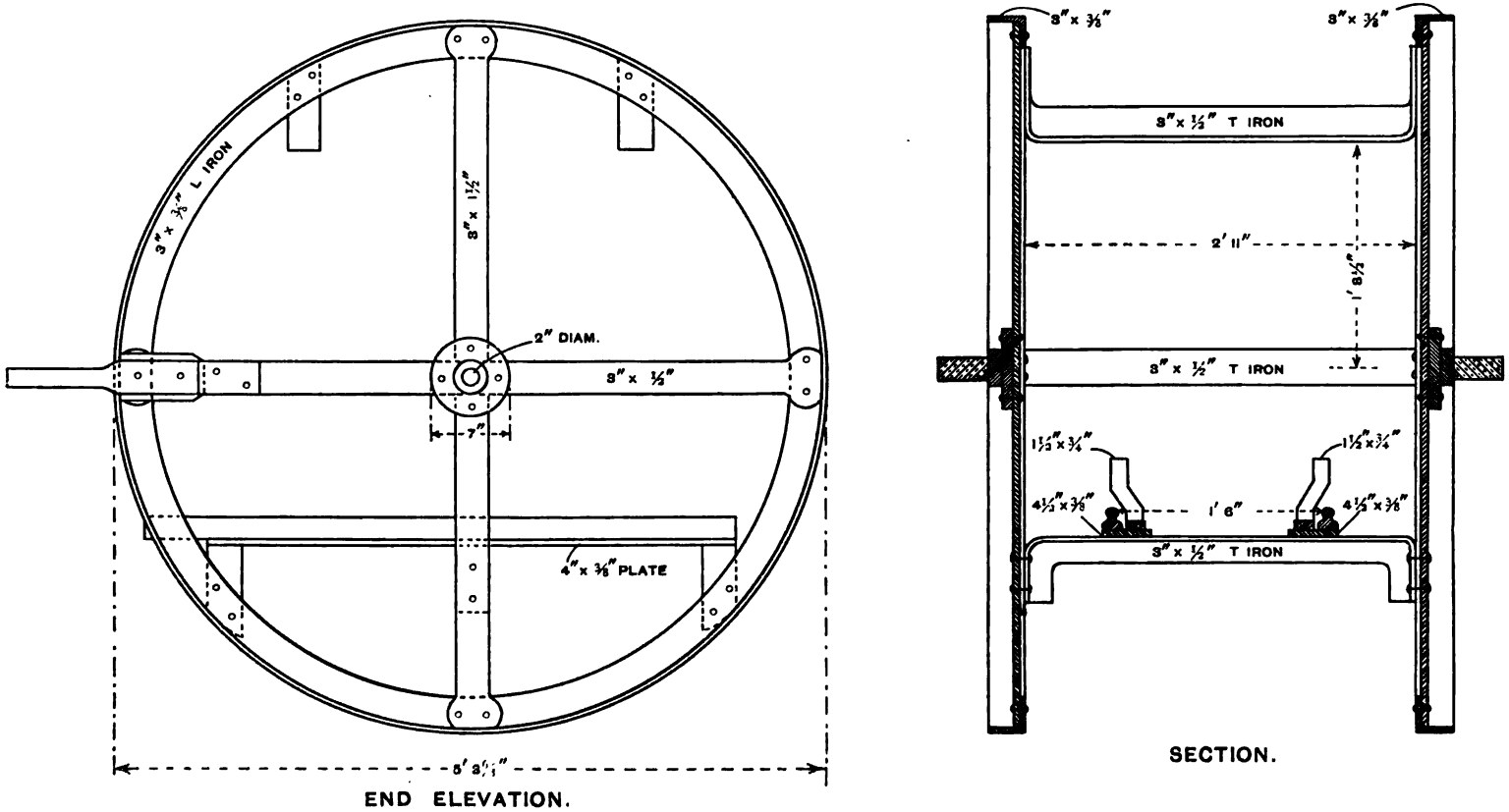


FIG. 103.—Tumbler used for tipping fixed trucks.

incline track for this purpose. By an arrangement of levers the two rails of a length of track are shifted, each a sufficient distance out of its position and away from the centre of the track, so that they serve as rails upon which the wide tread of the back wheels of the skip may mount. In their place, by the

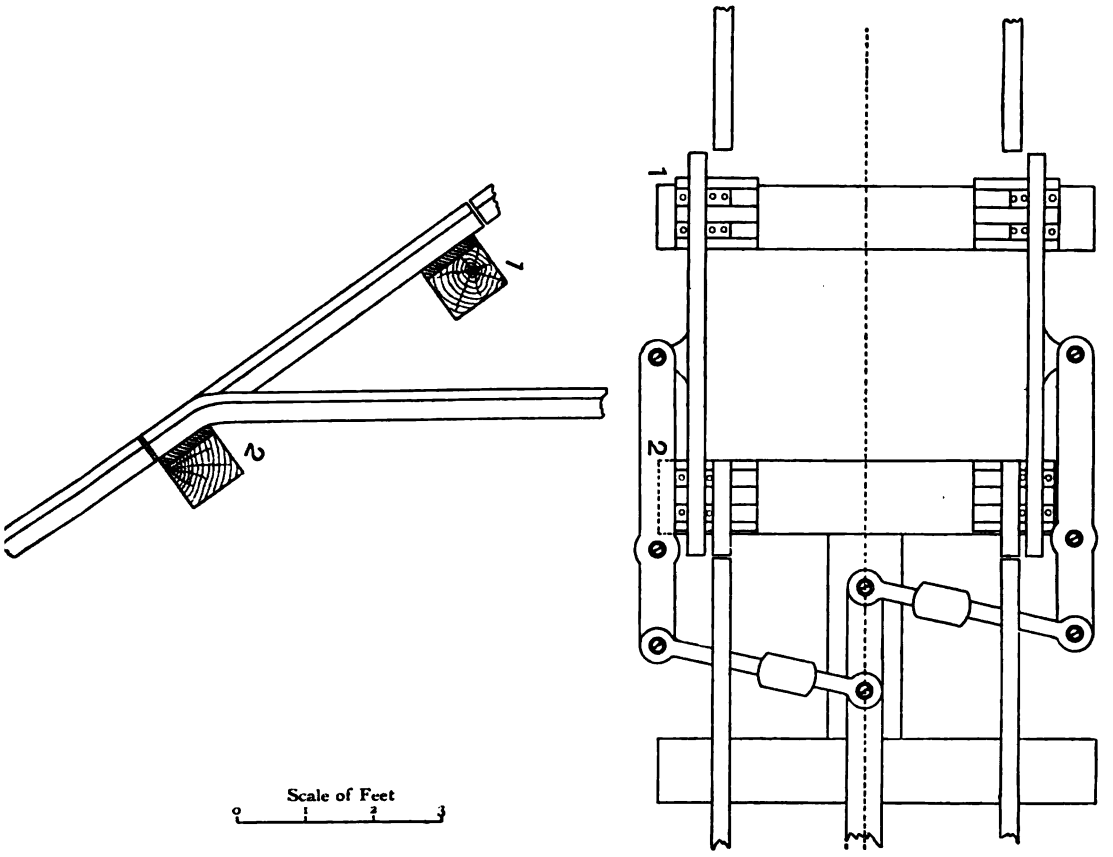


FIG. 104.—Method of temporarily breaking an incline track to provide a tip.

same movement of levers, two other rails slide, which at once curve to a horizontal position ; on these the front wheels run, so that the tipping is effected as with ore. In the Ferreira headgear, as shown in Fig. 94, a tumbler is arranged at the surface level to tip the drills. In some mines the drills are handed out of the skips, and in other mines they are taken up and dumped at the ore tip. In other cases a special

skip with an inclined bottom is used, which is so arranged that, after having been drawn to surface, the drills fall out when a door is opened; and again in other mines the skip, containing the drills, is taken off and landed by means of a gate.

It is always necessary to arrange that skips and cages can be taken off the track easily for repairs, etc. This is generally done, with incline shafts, by raising the skip a few feet above the surface level and by then shutting down a gate made of rails set at the correct gauge, so that when the skip is lowered it runs off from the shaft along these rails on to the horizontal level at the surface. These gates when not in use are suspended over the track out of the way of the skip, and their weight is so balanced that they can be easily let down or taken up. This arrangement is especially useful in connection with deep incline shafts where the men are lowered in proper trolleys, which when finished with have to be taken off. In vertical shafts with cages or skips, supported on frames which embrace guides, it is necessary, in order to take the skip off the track, to break the continuity of these guides for a length at the mouth of the shaft. This is done by hinging the guides to the uprights, to which they are attached above the collar set, so that they may be turned out of the way about vertical axes. The arrangement is shown in Fig. 105.

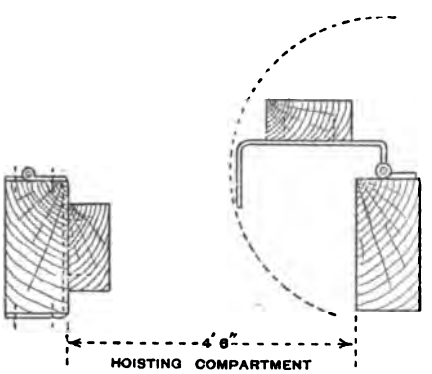


FIG. 105.—Method of removing the guides so that a skip carried on a frame can be taken off the winding track.

With skips on wheels in vertical shafts, the top or back guide of angle iron which is hinged with its continuation below the surface and connected with that higher up by bolts, is swung back; or it may be arranged that this portion of

the guide can be taken off altogether. After doing this, the skip can easily be hauled off the track.

Construction of Headgears.—As before stated, a headgear has to carry the rope and guide the skip or cage, so that the ore and waste may be landed at a height above the surface convenient for disposal.

The height of headgears is variously measured. Sometimes the actual vertical height of the uprights is taken, sometimes that between the pulley centre and the ground level, and at other times the vertical height over all the timbering, that is, from the level of the top platform to the level of the masonry at the base, is given. In the majority of cases these various distances are comparable, as the variations thus caused are inconsiderable. The usual height for an incline headgear is from 45 to 50 feet. Such a height provides ample headroom for the convenient conveyance of the ore to bins and of the waste to a dump. Where, however, sorting and crushing are done at the headgear a greater height is used. Thus those of the City and Suburban and Chimes West are 60 feet high, and that shown in Fig. 189 is 65 feet high.

The shafts of the first row of deep levels, which attain to vertical depths between 600 feet and 1600 feet, have headgears which are usually from 60 feet to 70 feet high, and those of the second row of deeper levels which have to go to greater vertical depths, have higher headgears, those of the Robinson Deep and Knights Central, for instance, being 85 feet high. A somewhat greater height is required with vertical headgears in order to give sufficient headroom for the landing-place at the mouth of the shaft.

The pulleys which carry the rope are carried on bearings at the top of the headgear. They are usually about 8 feet in diameter, though in the deeper levels they are larger, those of the Knights Central being 10 feet and those of the Robinson Deep 14 feet. With incline headgears the pulley is placed so that when the skip is tipping, and its lower end is raised above

its mouth, the rope is still inclined up to the pulley at a good angle. With vertical shafts it is placed so that the rope, as it extends to the skip at all positions in the shaft, is practically vertical. In the latter case, as the rope runs more or less in the centre of the shaft, the pulley is so placed that its rim runs tangential to the centre of the shaft.

In hoisting, the resultant of the weight in the shaft with the pull from the engine passes through the centre of the pulley, and it is this force principally which has to be resisted and allowed for in the construction of the headgear.

With incline shafts this resultant does not vary much from the perpendicular, so that the pulleys are supported either upon two main posts, more or less vertical, directly under their centres, as in Figs. 106, 189, or upon a platform of cross-pieces between four main posts, as in Fig. 95, which is the more usual practice. In the former case each vertical post is supported by a strut on either side, the front one going down to the mouth of the incline and the other extending back towards the engine-house. In the latter case, the back posts are placed at an inclination to act as struts, and two front struts, placed along the hauling plane, support the main posts on the other side. As in this case the resultant generally falls within the main posts, less strain in hauling comes on to the timbers which extend to the shaft-mouth. On these the hauling track up the headgear is laid. The main posts are generally arranged on a base sufficiently broad to make room for the ore bins, etc., and they are disposed symmetrically about the hauling compartments.

With vertical headgears the resultant falls on the engine side of the shaft and the headgear has to be especially stayed in this direction. It is invariably the practice in the deep levels to support the pulleys by cross-pieces between four posts placed symmetrically about the mouth of the shaft, including the pump-way, as shown in Figs. 89, 92, but sometimes in the shallower vertical shafts near the outcrop no

main posts are placed on that side of the shaft which is removed from the engine, in which case the pulleys are

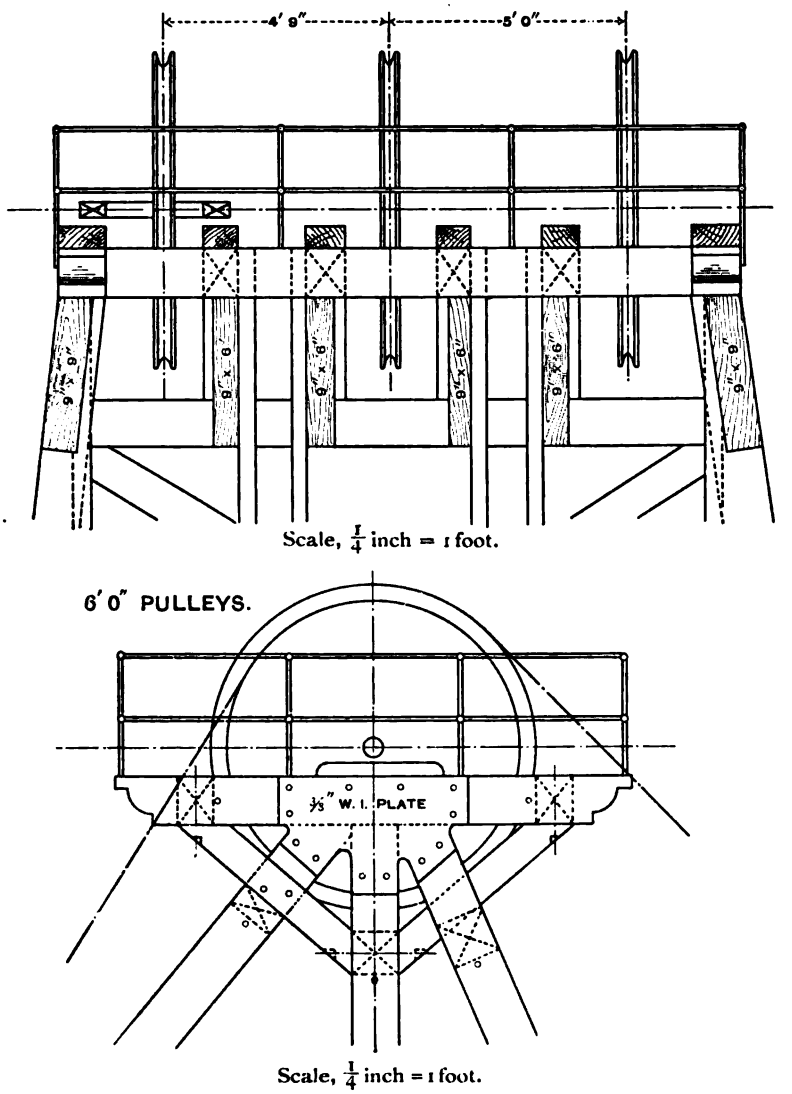


FIG. 106.—Support of pulleys on the Rietfontein A headgear.

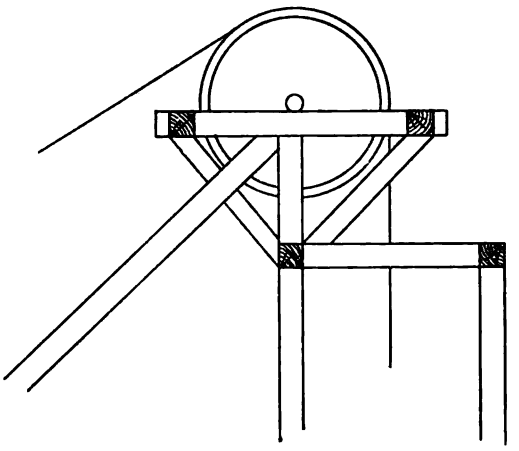
supported on a platform erected over two posts and built up from the two back struts, as shown in Figs. 94 and 107.

With vertical shafts the ore bins are entirely outside of the main timbers on the tipping side. Where there are four main

posts the two on the tipping side give some support to the bin structure, though this mainly rests on its own foundations.

With the single exception of those of the Robinson Deep, which are of steel, all the more important headgears are of timber, dimensions of which, in typical instances, are given in Figs. 89, 92, and 95. In these figures also, the nature of the foundations are shown.

It is almost invariably the case that at the tip the ore falls on to the waste door and then passes on to the grizzlies.



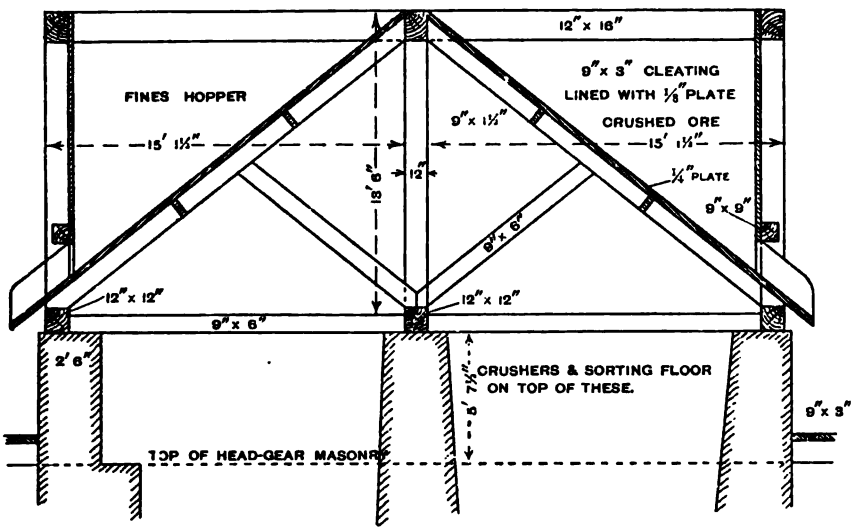
Scale, $\frac{1}{8}$ inch = 1 foot.

FIG. 107.—Headgear top, Langlaagte Royal.

When waste is being hoisted, this door, which is balanced so that it can be easily manipulated, is opened and the contents of the skip are discharged direct into the waste bin, as shown in Figs. 98, 100. The extent of grizzly surface depends upon the number of hoisting ways. They are usually set at an angle of about 40° and are about 12 feet long. The component bars are at such a distance apart as is in accordance with the closeness to which sorting is carried when sorting is done, and, where crushing alone is done, so that the ore which passes through shall not be larger than the size to which that which does not pass through is afterwards crushed. The usual space between the

bars is from one to two inches. These grizzlies are in all respects similar to those which are described under ore-dressing operations. It is noticeable that at the Chimes West headgear the ore is to some extent binned over the grizzly, although the general rule is to allow it to have a free fall.

There are generally three different bins at the headgear: one each for waste, fine ore, and coarse ore. In Fig. 91 the



ORE BINS, HEAD-GEAR. CHIMES WEST.
Scale, $\frac{1}{8}$ inch = 1 foot.

FIG. 108.—Ore bins at the Chimes West headgear.

ore is divided by screening into three products, so that, with one for the waste, there are four bins; but this is unusual.

The waste bin is usually high up on the headgear, so that its contents can be drawn off and dumped at a convenient level. Its capacity is not often large. The "fines" bin is in communication with the under side of the grizzly, and the coarse ore bin with the upper side.

The amount of ore which these two bins are made to hold between them depends upon the size of the property and the number of headgears. It varies from 100 tons up to

an estimated capacity of close on 1000 tons for the bins of No. 1 shaft of the Buffelsdoorn Estate, a usual amount in the larger mines being from 150 to 300 tons. They are usually set with the floors at an angle of about 40°. Their structure and the nature of their foundations are shown in Figs. 88, 91, 95, and 108.

The discharge from the bins is either through a sliding or a hinged door, and the falling rock is guided by passing

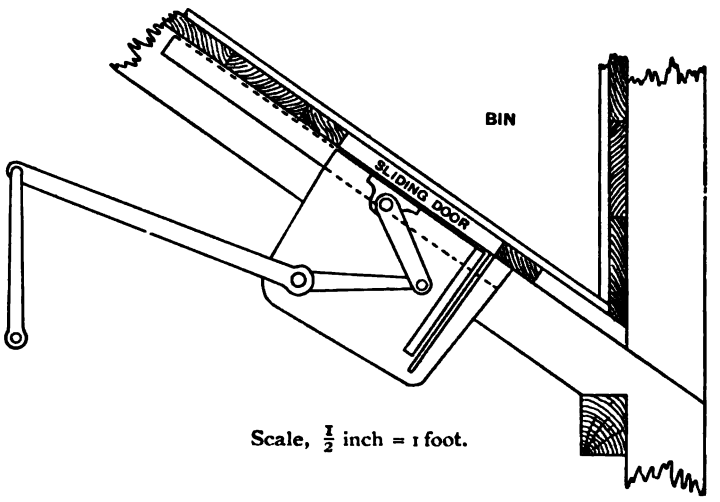


FIG. 109.—Arrangement for filling trucks from an overhanging bin-face.

through a shoot. The size of the opening in the bin-face is usually about 18 inches by 28 inches.

The sliding door is more generally used, and it opens and shuts by moving along the bin-face. The door itself is generally an iron or steel plate $\frac{5}{16}$ inch to ½ inch in thickness, which is guided on either side between two pieces of angle-iron long enough to allow the door to be fully opened.

The movement of the door is generally controlled by a lever with a connecting link, as shown in Fig. 110, but sometimes a rack and pinion are used.

When hinged doors are employed they are placed near the end of a fixed shoot. Though such doors are used alone, it is more usual to use them in conjunction with the sliding

door, as shown in Fig. 111; by this means, and by having only one of them open at a time, the discharge of the ore can

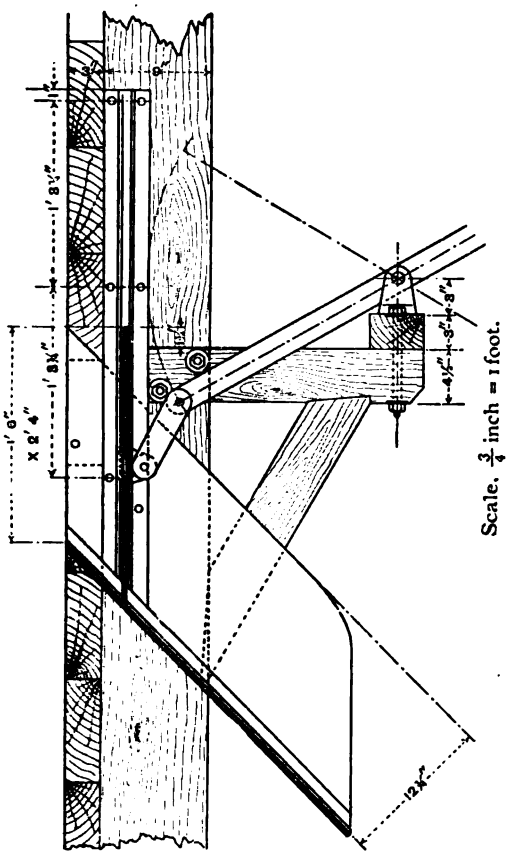
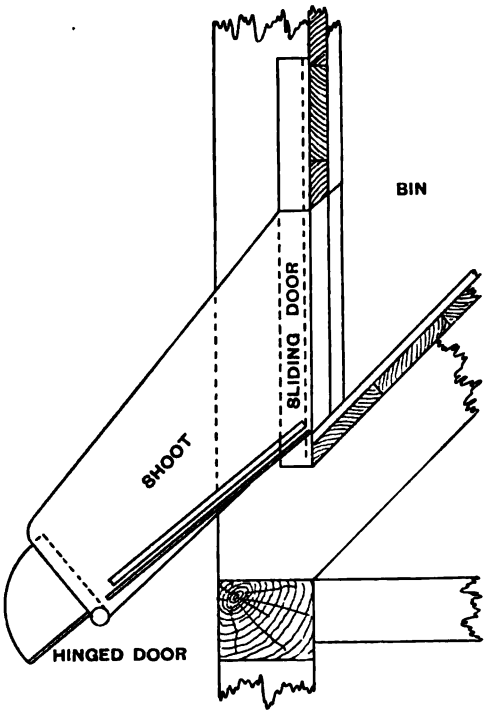


FIG. 110.—Sliding door for bin.



Scale, $\frac{1}{2}$ inch = 1 foot.

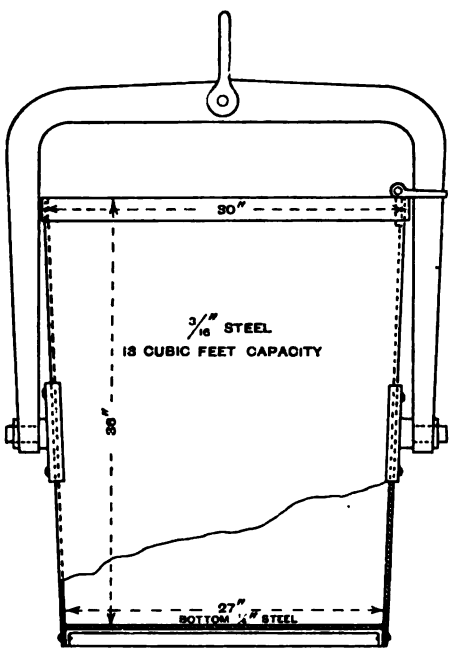
FIG. 111.—Arrangement for filling trucks from a vertical face of a bin.

be very well regulated. Fig. 109 shows the sliding door used when the trucks are run below the sloping bottom of the bin instead of in front of its vertical face.

CHAPTER XI

WINDING APPLIANCES

Skip, Cages, and Buckets.—Buckets are not used much in mining proper. In the early stages of shaft-sinking, when the broken rock is hoisted to the surface by windlasses and hand labour, small buckets or kibbles with a capacity of about 5 cubic feet are used. These buckets are suspended on trunnions from bows, so that in order to tip the contents it is necessary to lift the bucket till it is turned over, for which purpose a handle is provided on the bottom. When hoisting proceeds with steam, larger self-tipping buckets, the point of suspension of which is below the centre of gravity of the full bucket, are used. One of these is shown in Fig. 112. These are easily tipped by releasing a catch, which, when the bucket is in the shaft, keeps it from falling over. They usually hold about 20 cubic feet of rock. In order to keep these



Scale, $\frac{3}{4}$ inch = 1 foot.

FIG. 112. Tipping bucket.

buckets from swinging in the shaft the rope is guided and kept central by the frame shown in Fig. 113. This frame rides on a collar, fixed on to the rope just above the bucket, but it is not attached to either the rope or the bucket. At the bottom of the guides of the shaft timbering it is caught,

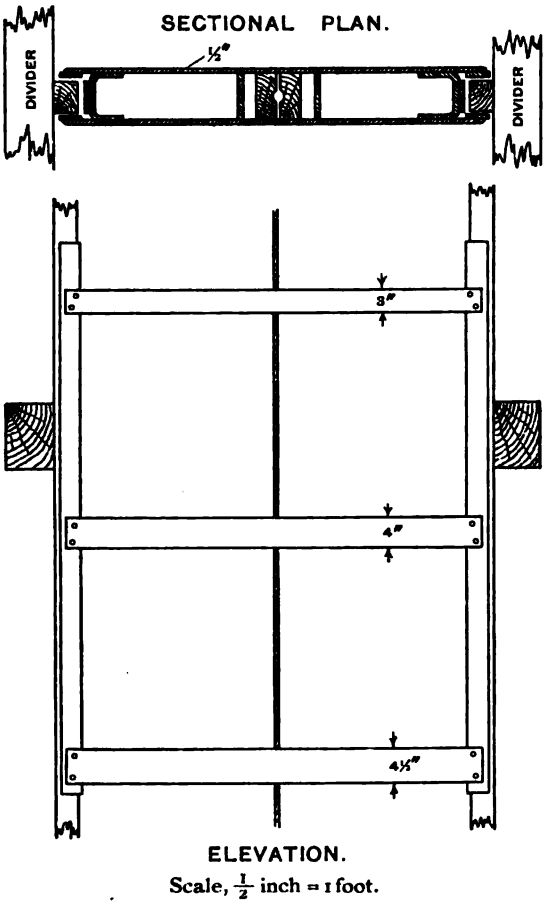


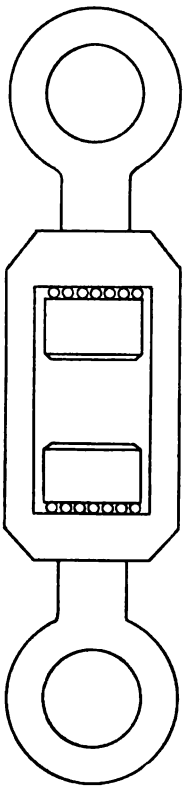
FIG. 113.—Frame for guiding sinking bucket. Approximate scale $\frac{1}{16}$.

whilst the bucket descends to the bottom of the shaft, the rope passing freely through the frame. When the bucket is again hoisted, as it enters the timbering, the frame is lifted and carried up again with the bucket.

There is a further advantage in using these buckets and frames for sinking purposes, in that the buckets can be

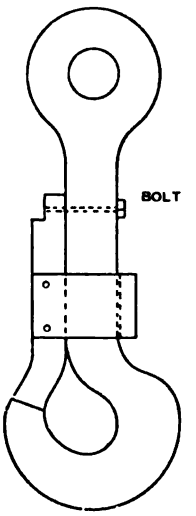
deposited at the bottom independently of the height of the last set of timber above the bottom, for the guide frame, not being connected with the bucket, is left at the last set. When, however, the shaft has been completed and timbered throughout, all the advantage is on the side of self-tipping skips, with the frame attached.

Another advantage in using a bucket for sinking purposes is that, with a detachable hook, such as is shown in Fig. 115, two buckets may be in use at one time for the same hoisting compartment, one being hoisted whilst the other is being filled at the bottom, a considerable saving in time being thus effected.



Scale, $\frac{1}{6}$

FIG. 114. — Swivel with ball bearings to take the torsion off a rope.



Scale, $\frac{1}{6}$

FIG. 115.—Detachable hook for a bucket.

The torsion of the rope is often sufficient in a deep shaft to cause a bucket to rotate rapidly. In order to lessen this tendency, a swivel with ball-bearings, such as is shown in Fig. 114, is sometimes used.

It is not usual to use buckets in sinking incline shafts, for the bottom is easily reached by wheeled skips, for which a track can be laid quite close down to the advancing face.

Skips.—All the skips used are self-dumping skips, that is to say, the discharge of the contents is effected quite automatically. An exception occurs, however, in the Jubilee mine, where the discharge is effected, as stated before, by opening a door hinged to the back of the skip, and towards which the bottom slopes at an angle of about 40°.

There are two kinds of skips in use: those which are

carried on frames running along the shaft guides, and those which run on wheels.

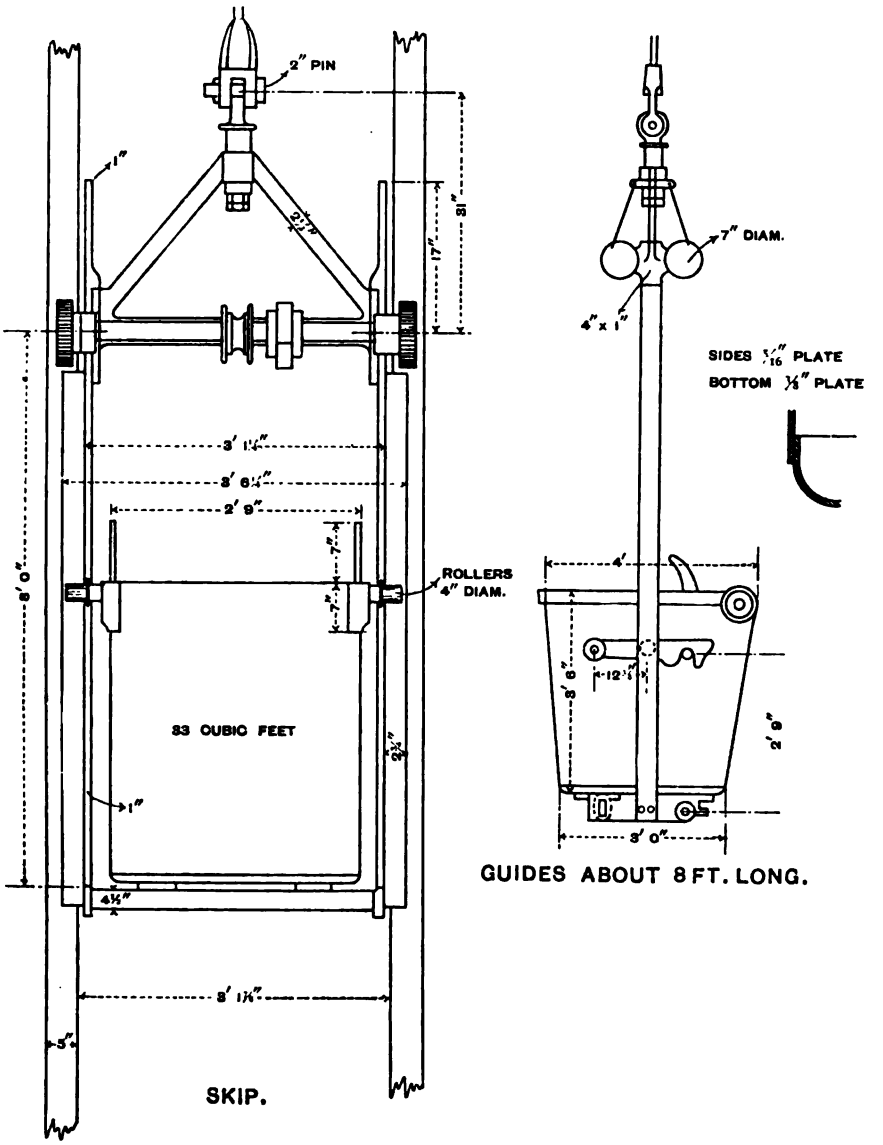


FIG. 116.—Skip for vertical shafts carried on a frame. (Robinson Deep.)

The former are at present only used in vertical shafts. One is shown in Fig. 116. The rectangular body of the skip, made of steel plates $\frac{3}{8}$ inch thick, is supported on the bottom

of the frame, to which it is also hinged on the tipping side. It has a capacity generally of about 30 cubic feet. A false bottom of inch planks is usually arranged in the bottom and covered with plates; the tipping is effected in the manner described on p. 232. From top to bottom and on either side there are iron runners fixed to the arms of the frame and channelled so as to embrace the guides of the shaft timbering. It is usually sufficient for the stability of the frame that these runners should extend for a length of about 8 feet; but in sinking, where the guides cannot be kept right down to the bottom, a greater length is used, depending upon the amount of headroom which there is at the mouth of the shaft. It is this consideration, together with the fact that this skip cannot be readily taken off the frame to be left at the bottom for filling, which somewhat limits its adaptability for sinking purposes.

The arms of the frame, at a height of about five feet above the top of the skip, are firmly bolted to the draw-bar in much the same manner as with the incline skip shown in Fig. 117, though the draw-bar itself is not so simple. It is shaped like an isosceles triangle, the base occupying the ordinary position of a draw-bar, and the other two sides meeting farther up the shaft. Through the apex so formed an eye-bolt passes, which has the rope attached to its upper end, and two chains in connection with four eccentric toothed wheels, one on either side of each guide, at the other end. As long as the weight of the skip is carried on this eye-bolt, the nut on its lower end is close up against the shoulder on the draw-bar, and the teeth of the wheels are restrained from embracing the guides; but when that weight is taken off, as happens when the frame is resting with its bottom on a platform, or when the rope is broken and the skip is in the shaft, these wheels, actuated by spiral springs arranged concentrically with them, revolve so as to grasp the guides, and the skip becomes held in the shaft. The rope is attached to the upper end of the eye-bolt by a shackle

and pin. This safety appliance is the one generally used to prevent the skip from falling down the shaft where there are wooden guides.

For running along an incline as well as in a vertical shaft, a skip carried on wheels is used. Such an one is shown

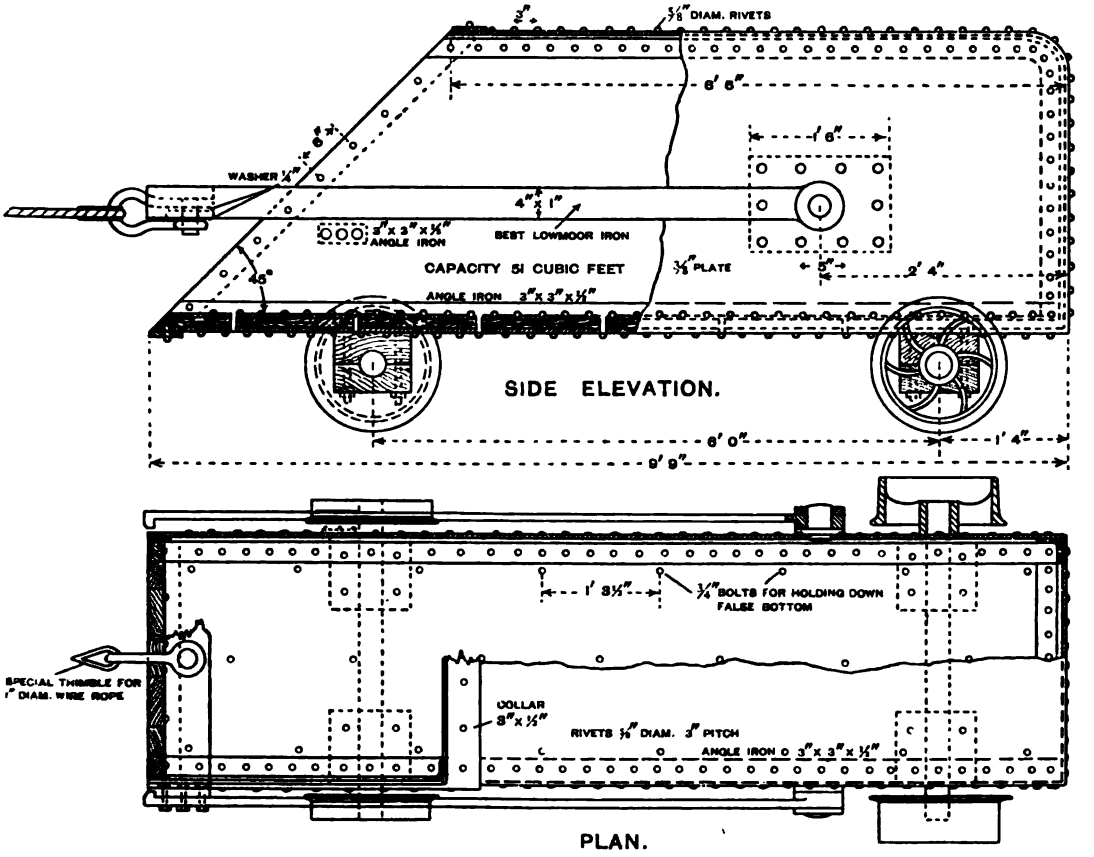


FIG. 117.—Wheeled skip for incline shafts.

in Fig. 117. Generally the body is made of $\frac{3}{8}$ inch steel plate riveted to lengths of angle-iron; the detail of the construction is well shown in the above figure. As these skips are designed only for running in shafts which at some portion of their depth are inclined, the back is cut away to facilitate filling, the amount depending upon the inclination of the shaft. At the Robinson, with an inclination in the lower

levels of less than 30°, the bottom plate is about 9 feet long, and the back one about 4 feet; the sides, however, are not cut away to the same extent. Generally the back plate at the mouth of the skip is set back from the bottom at an angle of 45°, or, where the shaft inclination is steeper, at an angle of 30°, the sides being cut away at the same angle. With the use of those shoots on the underground ore bins which face down the shafts, it is not necessary to cut away the back so much. Over the bottom plate a false bottom is sometimes placed, made of one-inch wood packing, covered with steel plates which are bolted through.

The capacity of the skip varies from 20 to 60 cubic feet or more. In the larger mines a load of 3 tons is generally carried. As a hoisting compartment is larger across than along a shaft, it is more usual to find that the height of a skip is greater than its breadth, common measurements being 3 feet and 2½ feet respectively. With most skips the wheels, of which there are two pairs, are placed on the bottom plate, at about equal distances from either end, but these positions are varied to suit the tip, as shown in Fig. 99. In the Ferreira and Jubilee mines they are centred on the side plates, and this necessitates raising the track up off the floor in the incline portion of the shaft, as indicated in Figs. 71, 72, whilst the wheels must, moreover, run loose on the trunnions. In this respect it is the better practice to fix the wheels on to axles by shrinking and keying, so that the axles may run in bearings which, affording a larger wearing surface than loose wheels on fixed axles, cause less wear and tear. These bearings are generally made of gun metal or hard wood. Again, where the wheels are centred on the side of a skip, they take up more room in the shaft, because they cannot be brought so close to the side of the skip, which is a disadvantage.

The wheels are usually made of cast iron, or sometimes of cast steel, with a diameter about 15 inches. For the

ordinary tip on the incline, it is necessary that the back wheels have a larger tread than the front ones, as shown in Fig. 117, in order that they may mount up upon the outside track, as explained on p. 230. The usual widths of tread are, back wheels $5\frac{1}{4}$ inches, front wheels $2\frac{1}{4}$ inches.

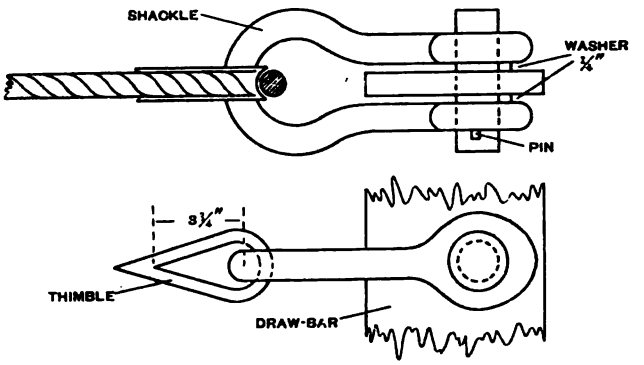
With other tips the wheels have all the same width of tread. With the arrangement shown in Fig. 99, a front pair of smaller wheels or rollers are required in addition to the four wheels.

Two arms called the draw-bar arms run outside the skip, connecting the skip trunnions with the draw-bar and thus taking the full weight of the skip. They are made of best malleable iron $\frac{3}{4}$ inch to 1 inch thick, and 3 inches to 4 inches wide, and are variously centred, often at the end, as indicated in Figs. 99 and 102, but also at all points on the lower quarter of the side plates, as in Fig. 117. In incline shafts, where the wheels have no guides to prevent their being lifted off the rails, it is perhaps better not to centre the arms at the end, because in that position, should an obstruction occur at the front wheels, the skip might easily be tipped in the shaft, but in hoisting in vertical shafts, there is an advantage in centring them at a bar which is fixed across the lower end, in that the weight is more securely supported. In tipping also, the success of some arrangements, as, for instance, that shown in Fig. 100, depends upon a direct pull at the bottom end. In all cases these arms swing freely over the back of the skip, but in the other direction they are prevented from coming on to the wheels by small stops of angle-iron.

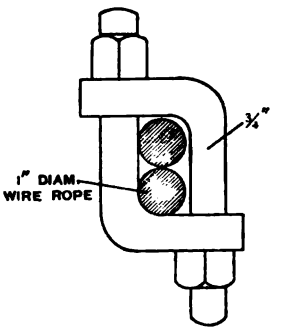
At the upper end they are attached, usually in a manner shown in Fig. 117, to the draw-bar, which is a cross-piece of malleable iron $\frac{3}{4}$ inch to 1 inch in thickness, and 4 inches to 6 inches in width. To this the rope is attached either directly with a shackle and pin, or by two chains, each with its shackle and pin. The rope is usually bent through the shackle, around a thimble, and turned back on itself for a length of 3 or 4 feet, being secured by such clips as are shown in Fig. 118.

For sinking purposes, when water has frequently to be hoisted, incline skips are generally provided with an inlet valve in the lower end, so that the water may easily fill into them.

With wheeled skips it is not usual to have any safety



Scale, 1½ inches = 1 foot.



Scale, 3 inches = 1 foot.

FIG. 118.—Shackle, thimble and clip for the attachment of the wire rope to the draw-bar.

appliances, but when men are being raised it is always required that the skip or cage shall not be wound at a speed exceeding 3.1 miles an hour after it has passed a point determined by the inspector of mines. Further, it is not allowed to hoist men in a shaft where the distance from the top landing-place to the underside of the headgear pulley is less than 26 feet. With these provisions, especially in incline shafts where the length of track from the shaft mouth to the tip is generally

several times greater than this distance, the chances of over-winding are minimised.

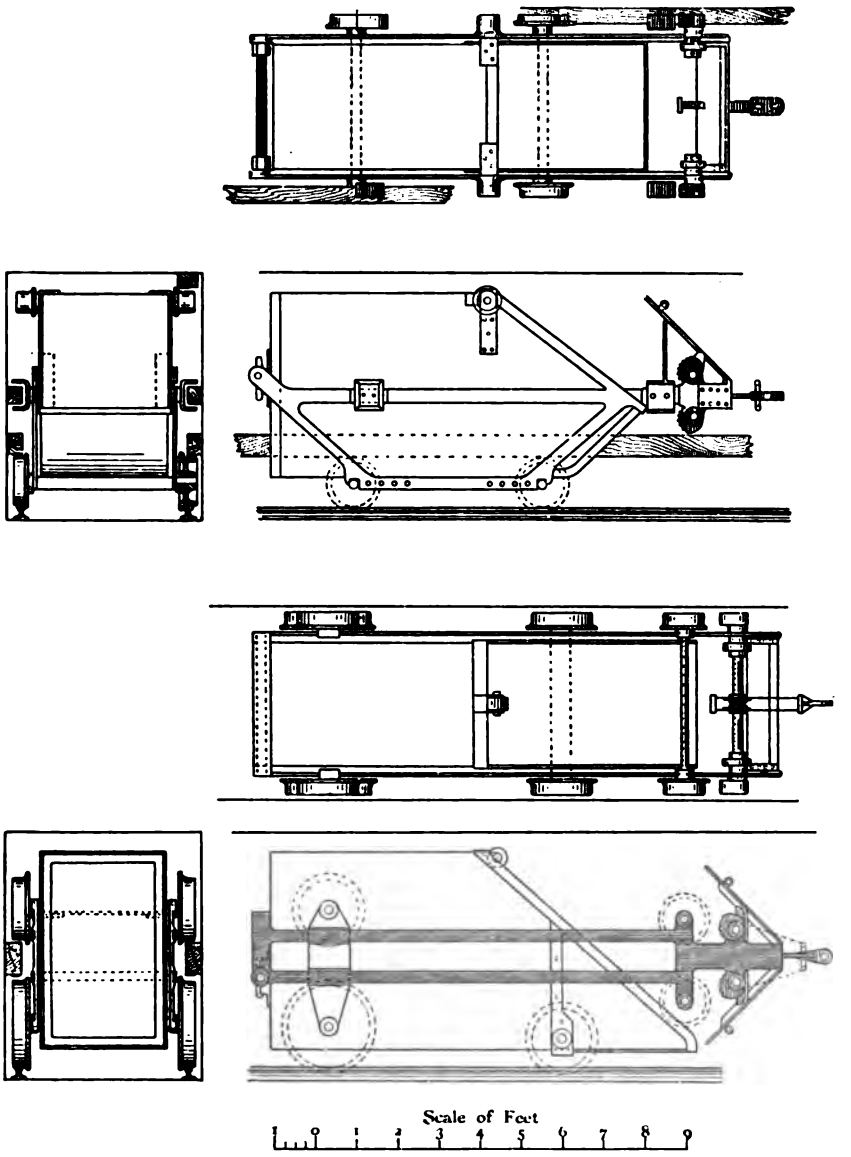


FIG. 119.—Pitchford's self-dumping skips.

When men are travelling in a skip which at any time runs in a shaft inclined at a greater angle than 45°, hoods or covers are placed over the skip to protect the men from

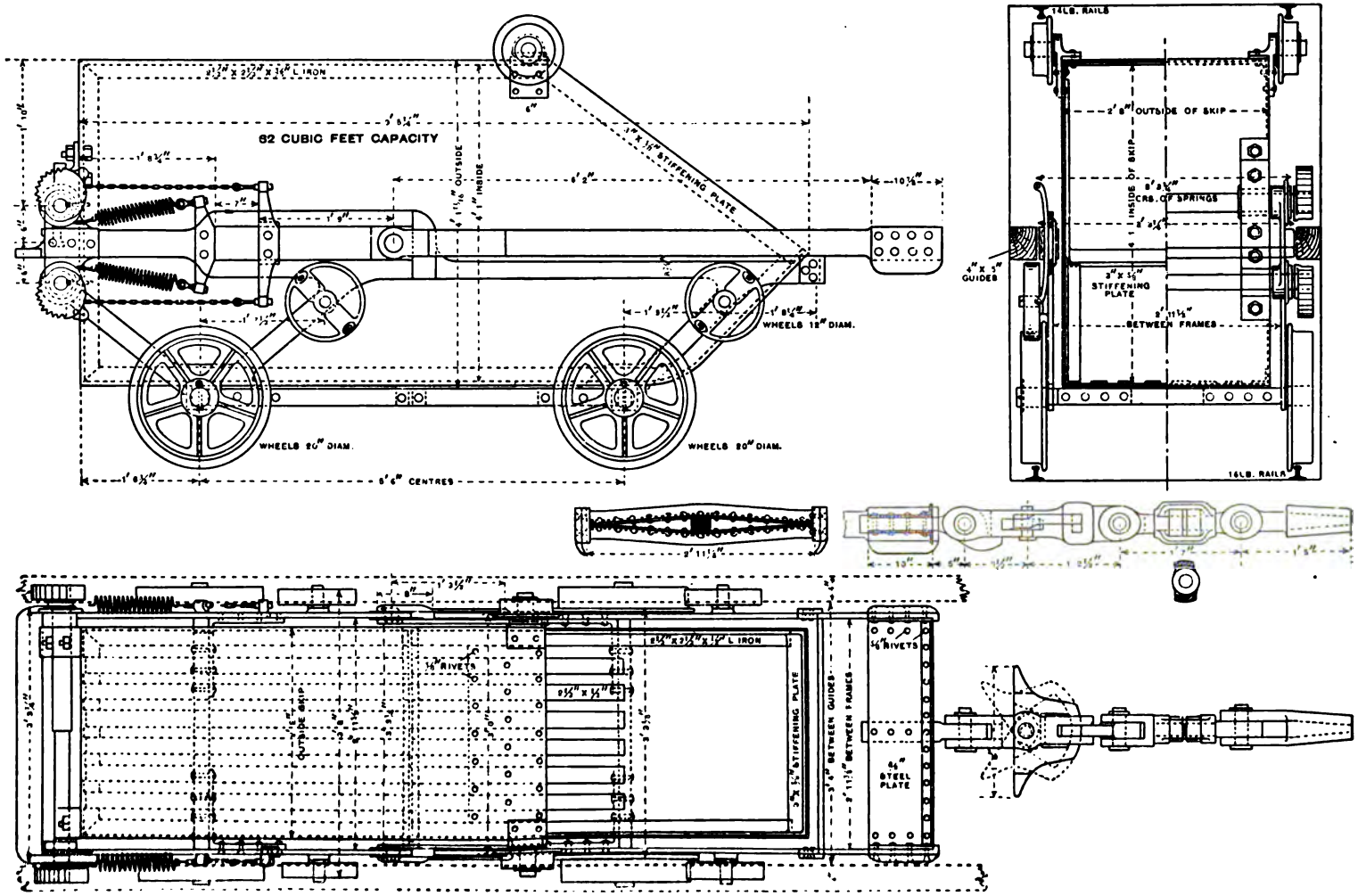


FIG. 120. Pitchford's self-dumping skip, arranged to tip backwards, and fitted with safety appliances against over-winding and rope-breaking.

anything falling down the shaft. These covers are variously made; sometimes they consist of two or more plates which

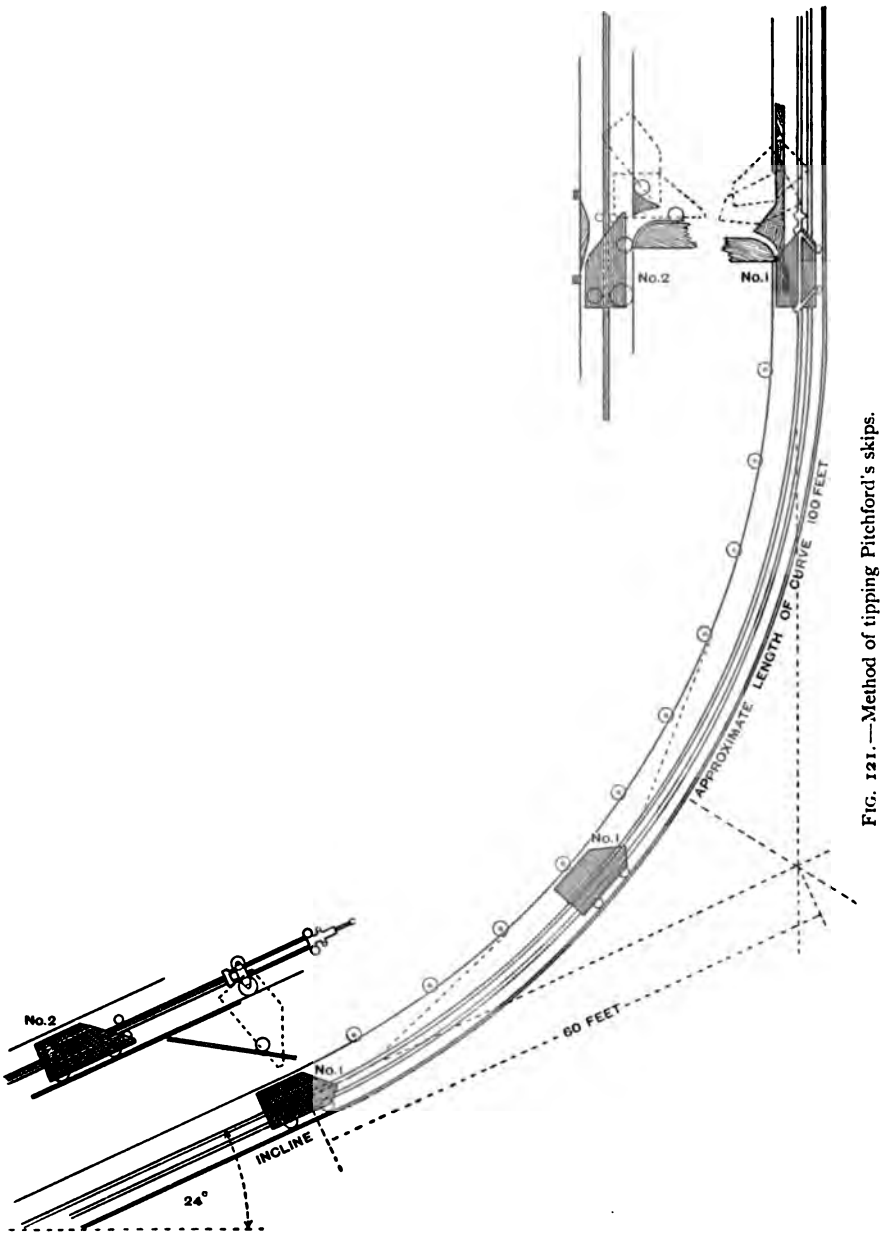


FIG. 121.—Method of tipping Pitchford's skips.

are hinged together and to the back of the skip, so that they can be folded up when not required, and secured to the back

of the skip; or they may be made as a curved hood, which is fixed and removed as desired.

Mr. Pitchford of Johannesburg, in his self-dumping skips for running in incline and vertical shafts, has applied all the safety appliances which are used with cages working in vertical shafts. In Figs. 119, 120, two of his skips are shown; of these

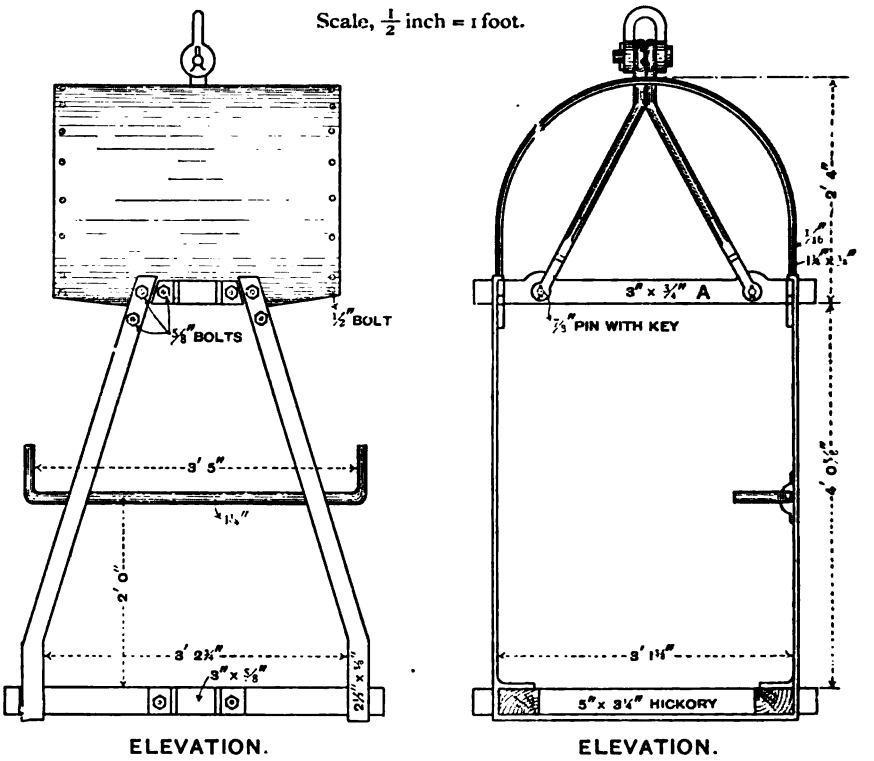


FIG. 122.—Cage to carry one truck in a vertical shaft.

that given in Fig. 120 is arranged to tip backwards, as No. 1 of Fig. 121, and the other—the lower one of Fig. 119—is arranged to tip in the ordinary way forward, as No. 2 in Fig. 121. The draw-bar arms are enlarged to constitute a frame upon which the body of the skip is to a great extent carried. This frame is fitted with wheels to run on the wooden guides —which for this purpose are continued from the vertical throughout the incline shaft—and also with other wheels

which run on the incline track. The method of gripping the guides in order to arrest the skip when the weight is taken off the draw-bar is identical with that in use with vertical skips, and the usual safety appliances in case of over-winding can also be attached, as in Fig. 120.

Cages.—Where ore is hoisted to the surface in trucks, cages are used. The great bulk of the ore is, however, hoisted in skips. Whilst in one or two cases larger cages are in use, it is the general practice to hoist one truck at a time in a cage such as is shown in Fig. 122. It is for lowering men that the cages are best adapted, and for this purpose it is likely that in the

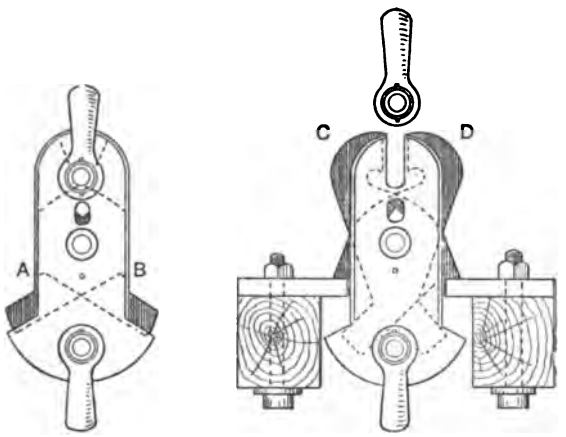


FIG. 123.—Patent safety detaching hooks for over-winding.

deeper levels double- and three-decked cages will be used; or with skips running on frames and provided with safety appliances, a cage may be attached to the bottom of the skip. It is usual to work these cages with some form of safety detaching hook, which can release the rope and thus prevent accident in case of over-winding; that which is in most general use is shown in Figs. 120, 123; the left-hand drawing of the figure above shows the hook in its normal position, the right-hand one showing how when wound too far the hook enters a ring which forces the blades C and D of the hook apart, and thus releases the shackle to which the rope is secured.

For carrying drills in cages a good arrangement is to let

a well into the centre of the floor, of such a depth and diameter that the largest drill, inclined at the greatest possible angle which the well permits, does not project outside of the cage.

Dogs and Gates.—Where cages are used, they are brought to rest at each station on supports called "dogs" (answering to the "keps" of the English miner), which are placed so that the floor of the cage is on a level with that of the station, and that the trucks may be easily run on or off. These dogs are made and manipulated as shown in Fig. 124. In incline shafts the skip is sometimes brought to rest against a support whilst it is being filled at the loading station. In some mines a heavy chain is arranged to hook across the shaft for this purpose, and in others, suitable gates of timber, properly counterpoised, are used. In Fig. 76 a gate of timber, hinged near the roof of the shaft to a piece of timber, 9 inches by 9 inches, is indicated.

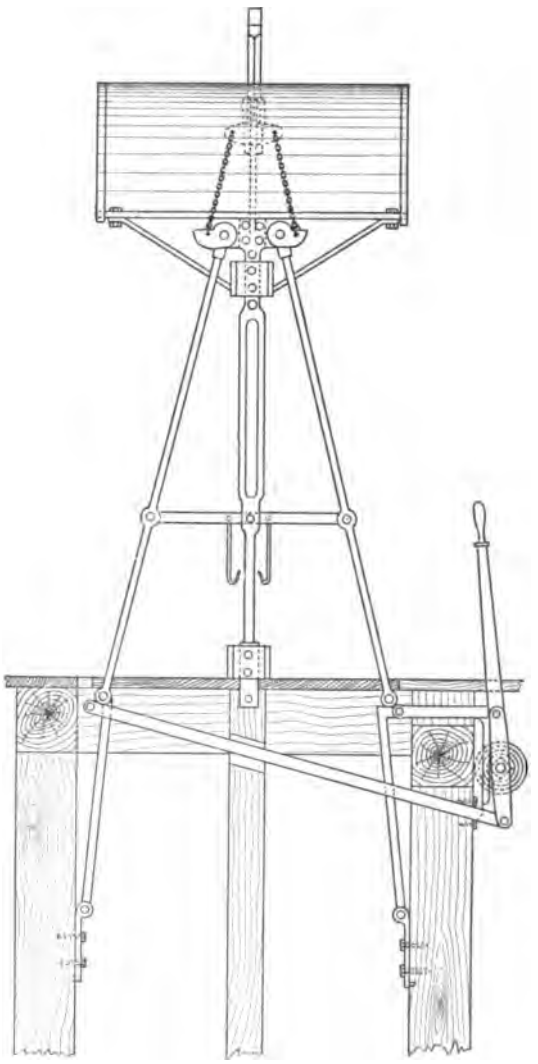


FIG. 124.—"Dogs" for supporting cages at landing stations.

With shafts which are steeply inclined or vertical the

entrances of the compartments at all stations, including that at the surface, are barred by gates which are only opened when the skip is there. The usual style of gate is the ordinary hinged one. Sometimes at the surface, when cages alone are used, the gate is so arranged as to be lifted by the cage as it arrives, and again let down into its place as the cage descends. Where a skip comes to the surface through a vertical shaft it is sometimes arranged that it may be supported on dogs so that its lip is on a level with the surface, in order to fill in drills, etc., for conveyance underground.

Rails, Guides, Pulleys, etc.—In incline shafts the rails are usually similar in section to railway metal, and, where heavier skips are used, they vary in weight from 40 to 45 lbs. per yard. They are fixed to the sole pieces of the shaft timbering by large dog spikes, and are laid to about a 3-foot gauge. In the Jubilee and some other mines the incline track is made of lengths of flat iron screwed on to square timbers which run longitudinally under them.

Along the track the rope is kept from cutting into the sole pieces by cast-iron rollers placed at regular intervals.

The wooden guides for vertical shafts have been described under timbering. Where wheeled skips are in use, these guides are taken out and the skip wheels are guided between a continuation of the inclined track with lighter rails (30 lbs.) on one side, and an additional track of angle iron on the other. These two guides are placed at such a distance apart that, with the wheels running on only one track at a time, the skip is allowed but little play. At the turn from the incline to the vertical, in order to afford an easy entrance for the wheels, the upper guide of angle iron is more widely separated from the main track, but at a few feet from the entrance it is brought down to its correct distance. In order to guide the rope around this angle, it is carried on pulleys, centred so as to keep it off the inside of the curve. As in hoisting, the rope is likely to shift its plane, more than one pulley is required

in each compartment. It is better to have four or five running loose on an axle and made to occupy the centre of the compartment by a restraining collar on either side.

Where trucks are hauled directly up the incline, a length of track is arranged at each station as a gate, so that when it is lowered the trucks may either be run off from, or on to the station. Where the reef is flat enough, as it is at the Nigel mine, the incline track is directly connected up by points with the horizontal track at some levels. In these cases, as the load on each wheel is not heavy and the hauling is done slowly, the incline track is not much heavier than the horizontal track, the metal rails weighing from 16 to 20 lbs. per yard.

Ropes.—It is required by law that for raising or lowering persons, the winding ropes shall be of steel, and that they must permanently possess a breaking strain of at least six times the maximum load they are required to carry; in consequence, steel-wire ropes are invariably used. For hoisting, these are generally made of six strands, each containing about nineteen wires; the larger the number of component wires making up a given diameter, the greater the flexibility of the rope. The core is either of hemp or of steel wire. It is usual to consider the safe working load of such ropes to be one-fifth the actual breaking strain; for a 3-ton skip, which, loaded, weighs about $4\frac{1}{4}$ tons, a rope one inch in diameter is used in an incline shaft (see Appendix I., p. 480).

Flat ropes are made of several round ones, laid side by side and bound together by strands of charcoal iron or very mild steel. Generally round ropes are preferred to these, and especially where continuous hoisting through incline and vertical shafts is adopted, because the wear and tear on flat ropes around the sheaves at the curve would be very great. They have been adopted in some of the vertical shafts; at the Robinson Deep No. 2 shaft, the rope is 6 inches flat with a thickness tapering from $\frac{7}{8}$ inch to $\frac{3}{4}$ inch. In very deep shafts flat ropes present this advantage that they are more readily

tapered than round ones, and by using a taper rope the total weight to be carried is less than with a parallel rope, whilst the cross-section of the taper rope can, at every point of its length, be made sufficiently large to carry the weight of the loaded skip, together with that of the rope below it. Theoretically speaking, there is a limit to the depth of shafts in which parallel ropes can be used, but none when tapered ropes are employed.

Winding Engines.—The general type of engine employed is a simple, direct-acting, non-condensing, double-drum hoist.

There are very few compound engines in use; the direct-acting engine is preferred to the geared, because at high speeds there is less wear and tear, and also because, with no reducing pinion between them, a drum of less diameter gives as great a speed as a larger drum with a geared engine, and the load acts at a smaller leverage, so that, after starting, the full speed of the engine is more quickly reached. There are very few condensing engines because the work of hoisting is more or less intermittent, whereas a condenser should work regularly, and the winding engines are generally distant from any station where central condensation might be carried on. A double-drum winding engine is preferable to one with a single drum, for all the advantages of a single drum, such as balanced working, can be obtained, and, in addition, each drum can be worked independently if required.

In direct-acting double-drum engines the two drums run loose on the same crank shaft and are thrown into gear by friction clutches; when both are in gear, one is lowering whilst the other is hoisting, as one rope is wound over the drum whilst the other is wound under its drum in the reverse direction; when one has been thrown out, the other can be used as an independent engine.

The drums for round ropes are smooth cylinders, about 8 feet in diameter and 5 feet wide between the sides, which are a foot or so deeper all round than the tread; the drum is

generally large enough to take the whole length of the rope without overlapping, which would cause undue wear of the rope. For flat ropes reels are used with a minimum diameter of about 4 feet, and so narrow that the rope of necessity coils upon itself, increasing with each coil the effective diameter of the reel. On one side of a drum there is a brake rim, upon which post or bar brakes act; these are actuated either by steam or by hand. They should be sufficiently powerful to hold the hoist against the full head of steam; further brake power is afforded by a band working upon the circumference of the crank disc.

When a drum is working independently it is usual to lower by means of the brake, but where two drums are balanced, one is used to lower whilst the other is hoisting. It is advantageous to run thus, because the weight of the receptacle on one rope is always balanced by that on the other, and, on an average, about half the weight of the rope in addition, so that with the same engine a greater load may be carried and less power wasted.

The engine is usually arranged with twin cylinders, one working at each end of the crank shaft; the size of these and the steam pressure employed depend upon the load which has to be hoisted, and, as the weight of the rope is considerable in deep-level mines, also upon the depth from which hoisting has to be done. The winding engines working in incline shafts have, as a usual size, cylinders about 16 inches in diameter with 48-inch stroke, and the steam pressure is generally from 80 to 100 lbs. per square inch; with these, an average speed of about 1000 feet per minute with a full load of 5 tons can be maintained in shafts at an average inclination; but in the deeper vertical shafts larger cylinders and greater pressures are necessary.

At the Jumpers Deep No. 1 shaft, which has one sinking and two main hoisting compartments, there are two direct-acting engines; one is a double-drum hoist with two twin

cylinders 18 inches diameter and 48 inches stroke, 22 feet apart and steam-jacketed; it is fitted with two vertical cylinder steam post brakes, and with outside friction clutches and Corliss gear. The drums are 8 feet in diameter, carrying 2000 feet of rope, and the engines are designed to hoist a load of 6000 lbs. from a depth of 2000 feet at a rope speed of 1700 feet per minute, the steam pressure being 120 lbs. per square inch.

The other hoist is a twin-cylinder, direct-acting, single-drum hoist of about 340 horse power; the cylinders are each 20 inches in diameter with 48-inch stroke, and they work at a similar pressure to that mentioned above. With a pressure of 140 lbs. this hoist could exert over 400 horse power, capable of raising a load of 5 tons from a depth of 2000 feet vertical, at a speed of 1500 feet per minute. The drum is 8 feet in diameter and 6 feet wide; it is fitted with post brakes; Corliss gear and steam reversing gear are fitted to the cylinders.

At No. 2 shaft, Robinson Deep, a direct-acting double-reel hoist of 600 horse power has been erected. The twin cylinders are 24 inches in diameter and 72-inch stroke; they are fitted with steam reversing gear and Corliss valves. The reels have a minimum diameter of 5 feet, and they are adapted to receive a 6-inch flat rope. They are fitted with heavy post brakes actuated by steam, and further brake power is obtained by a band on the crank disc. This engine will hoist the ore up to surface through the vertical shaft from a depth of 1877 feet.

A further advantage of using a double-drum hoist is that the ropes can easily be adjusted to suit the different levels from which hoisting is proceeding, because, as each drum can be worked independently, its rope can be wound or unwound as desired, whereas with a single drum one rope, or one end of the rope, is being unwound whilst the other is being wound up, and to alter the working lengths of the rope entails con-

siderable labour. A device known as Whiting's hoist has been arranged to overcome this difficulty and to permit the use of small drums and round ropes; it consists in passing the rope around two grooved drums and balancing the cages, so that the rope only passes a few times around the drum. The following description of this hoist is taken from Mr. McDermott's paper, "Hoisting from Deep Shafts," read before the Institute of Mining and Metallurgy, 17th June 1896.

"It will be noticed that the hoist consists of twin engines, driving direct on a small grooved drum, which is coupled by two connecting rods to a similar drum in front, like the two driving wheels of a locomotive. The front drum is set at a slight angle to the horizontal, so that the rope may lead straight to the proper groove of the other drum and cause no chafing. The rope, after several turns over both drums, leads back behind the hoisting engine and around an anchorage grooved pulley fixed on a tension carriage, which runs on a trail track, and is adjustable, as to distance from hoist, by a steam winch. This tension carriage controls, by its position, the depth from which hoisting is to take place from time to time. It is fitted with grips which clamp the carriage to the rails while hoisting is in progress, and which are released when a new adjustment of the depth of hoisting is necessary. The whole operation of changing the adjustment from one level of the mine to another occupies only a few minutes. This adjustability is very convenient for sinking work, as the skip may be run down to any point desired, while the upper one is just on the landing dogs. Any stretching of the rope can be immediately remedied by the tension carriage adjustment."

It is proposed in the deeper mines, where both the vertical depth to the reef and afterwards the inclined depth on the reef are great, to use a separate hoist at the head of the incline just below the bottom of the vertical. As steam is unsuited for transmission to a considerable depth down a shaft, these hoists will probably be driven by electricity or perhaps com-

pressed air. At the Robinson Deep No. 2 shaft an electric hoist of 350 horse power is to be placed in this position and to work at 110 volts and 280 amperes, the power to be transmitted down the shaft through a $2\frac{1}{4}$-inch cable at a voltage of 2300. This electric hoist is indicated in Fig. 74, as it will be when in position.

In No. 2 shaft, Nourse Deep, the vertical portion of which has a depth of about 1500 feet, an air hoist will be used to haul upon the incline, though in No. 1, with a vertical depth of about 985 feet, continuous hoisting through both incline and vertical portions has been adopted.

Both the electric and air hoists, except in the means by which the power is transmitted and converted, are identical with the steam hoists above described.

It is required by law that every engine used in raising or lowering workmen shall be provided with a depth indicator, in addition to any marks on the rope, which will clearly and accurately show to the engine-driver at his driving seat the position at all times of the cage or skip in the shaft.

For this purpose, where the shafts are not too deep, a dial is used with a circular face, around which a radial pointer travels. The movements of the drum must be so reduced by worm, mitre or bevel wheels, that when the load is at its deepest the pointer shall not have made more than one complete revolution.

In winding from deeper shafts, a contrivance which allows the depth to be represented at the surface by a comparatively great length is obtained by marking the stations on a spiral groove running round a cylinder; to this cylinder the reduced movement of the drum is communicated so that it slowly revolves, and in doing so causes a pointer, which is suitably fixed in front, to move up and down as the cage rises or falls.

It is also required that in shafts exceeding 100 metres in depth the indicator shall ring a bell in the engine-room when the load is 20 metres from the top landing-place.

Every winding shaft must be provided with some proper means of interchanging distinct and definite signals between the top and bottom of the shaft and the intermediate stations, and, where the conditions require it, the bank of the shaft is connected by similar signals with the engine-room. These signals are knocks or rings. The knocks are made by allowing a hammer to strike against a loosely-placed iron plate, the hammer being worked by means of pulls upon a wire rope which passes down the shaft. This method of signalling has now to a very great extent been superseded by electric bells.

The following shaft signals have been prescribed by the Government :—

(1) Raise, when engine at rest.
(1) Stop, when engine in motion.
(2) Lower.
(3) Men about to ascend or descend.
(3) (In reply) Men may enter the cage or other conveyance.

Other special signals determined upon by the mine officials are also in use.

Considerations in Winding.—Ore and waste have at all times to be kept separate, and to be brought to the surface in such a way as to be readily disposed of. This is best done by placing the ore in bins, and by running the waste off at a level convenient for dumping.

For this purpose self-tipping skips are much more effective and speedy than cages and trucks, as these latter require to be handled on the surface by banksmen, and, in addition, when for any reason the hoisting is stopped, the tramming underground is similarly affected. With skips, however, and sufficient bin capacity, this tramming underground is rendered to a great extent independent of the hoisting. Again, the dead weight which is carried in the cage is, for the same weight of ore hauled, much greater than with the skip, because

the ore has to go up in trucks, so that there is the combined weight of the cage and trucks against that of the skip.

There is no more wear and tear with skips than with cages except at the tip, so that, considering the extra amount of dead weight to be hoisted and the time lost both in manipulating and waiting, it seems probable that hoisting in cages and trucks should be more expensive than hoisting in skips. There is nothing in the nature of the ore which makes it necessary that it be hoisted in trucks, nor is there any advantage gained by so doing. These comparisons have been made under the premise that a cage is a contrivance which, whether in an incline or a vertical shaft, provides a platform on which trucks may be placed to be raised. At the West Rand Mines an incline cage or platform is in use, on which trucks are hauled to the surface.

Where continuous hoisting is done from the incline on to the vertical a skip alone can be adapted to suit the conditions in each shaft. Cages are in all cases more suitable and safer for winding men, and in the deeper shafts it is likely that they will be used for this purpose.

It is intended in the deeper levels to sink vertical shafts down to the reef, and then to sink incline shafts along the plane of the reef, and in some of the deep levels this has already been accomplished. This system necessitates hoisting in both vertical and incline shafts.

There are two methods of doing this—

(1) By continuous hoisting through both shafts.

(2) By separate hoisting in the vertical and in the incline shaft.

Within certain limits continuous hoisting is the more suitable as being the less expensive. It is arranged for by means of suitable timbering at the angle connection between the two shafts, such as is shown in Figs. 65, 67, which allows the wheeled skip that runs up the incline to be also guided up the vertical shaft.

When, however, a vertical shaft is so deep that, owing to the longer time occupied in hoisting, it is calculated that a sufficient amount of ore could not be brought to surface by continuous hoisting, it is necessary to hoist up the incline with a separate equipment, which will place the ore in bins ready for discharge into the skips working in the vertical shaft, as shown in Fig. 74. Where these bins are so large that no delay is caused in waiting for ore, practically double as much can be hauled by this system as by the previous one. The exact point at which this system becomes the best is a matter for calculation.

The speed at which continuous hoisting can be done is somewhat limited by the angle connection, where there is considerable wear and tear on the rope. It remains to be seen whether the parabolic curve shown in Fig. 67 allows of any considerable increase of this speed in practice. It is likely that the separate system will prove the safer, because each shaft will be worked with an equipment which will be perfectly suitable to it, and from this it also follows that higher rates of running can be maintained.

So far, continuous hoisting has been adopted to a great extent in the first row of deep levels, where the vertical shafts are about 1100 feet deep, and where the incline shafts will ultimately be about 4000 feet in length.

As stated elsewhere, it has been adopted in the Nourse Deep, No. 1 shaft, where the vertical shaft is 985 feet deep; but in the No. 2 shaft of the same mine, about 1440 feet deep, separate hoisting has been arranged for the incline.

In No. 2 shaft of the Robinson Deep, the vertical portion being 1877 feet deep to the reef, separate hoisting has also been adopted, and this is the first of the second row of deep-level shafts to be equipped.

Costs of Hoisting.—In the majority of mines the expenses of hoisting and pumping are lumped together and calculated upon the unit, the ton of ore milled, so that the actual expenses per ton hoisted are difficult to obtain.

From the General Manager's Report, 31st December 1896, the following figures obtained at the Robinson mine were taken :—

For 11,193 feet of drives, winzes, and cross-cuts made during the year ending 31st December 1896 the following expenses were incurred in hoisting :—

	£	s.	d.
Engineers, banksmen, and bell tenders	704	13	6
Natives and food	119	11	0
Engine and banksmen's stores	142	9	5
Fuel	813	10	4
	£1780	4	3

This is equivalent to 3s. 2.17d. per foot of development, or about 10.9d. per ton hoisted. This development took place at an average vertical depth of about 725 feet below surface.

The following statement gives some costs which have been taken from the published Reports of the Companies mentioned :—

COSTS OF HOISTING AND PUMPING, INCLUDING WORKING COSTS AND MAINTENANCE, CALCULATED UPON THE UNIT OF ONE TON MILLED

		Cost per ton milled. s.	d.
Geldenhuis Estate, incline with skips, year ending 31st March 1897		0	8.59
Crown Reef, incline with skips, year ending 31st March 1897—			
	d.		
Banking and hoisting { Working costs	8.342		
Banking and hoisting { Maintenance	0.463		
	8.805		
Pumping { Working costs	3.917		
Pumping { Maintenance	3.947		
	7.864	1	4.669
Simmer and Jack, chiefly incline but partly vertical, with skips, 1st March to 31st December 1896		1	5.094
New Primrose, vertical and incline, cages and skips, six months ending 31st December 1896		2	0.24
Jumpers, vertical and incline, cages and skips, year ending 31st July 1897		2	3.42

In the case of the Crown Reef the hoisting expenses are kept separate from those of pumping, so that the figures given for this mine are more useful; they are particularly interesting as showing the low cost of maintenance in hoisting. As during that year little sorting was done, the only correction which has to be made is to allow that in addition to the number of tons milled a certain amount of waste has been hoisted; this correction will probably bring the cost of hoisting per ton drawn to about eightpence.

The figures given in the foregoing statement are not on a suitable basis for comparison, because of the various amounts of waste hauled, and of the different quantities of water pumped. Though this be the case, there is yet sufficient evidence in the figures to show that continuous hoisting on the incline and in skips is cheaper than hoisting in stages, partly in cages in vertical shafts, and partly in skips on incline shafts.

CHAPTER XII

PUMPS AND PUMPING

THE amounts of water met with in the mines along the Witwatersrand are small when compared with mining districts in other countries. Mr. William Hall, in his evidence before the Industrial Commission, June 1897, stated that for the 15 miles of Central Rand the ruling quantities of water were as follows:—

"(1) The outcrop companies yield an average of 50,000 gallons per shaft per day, the range being from 10,000 to 90,000, and there being three or four cases where the amount runs materially higher.

"(2) That the first row of deep levels as a rule yield about 45,000 gallons per shaft per day, the range of amount being from 8000 to 80,000 gallons, and there being two or three cases of very materially higher water output; and

"(3) That the second row of deep levels, with one exception, yield from 2500 to 5000 gallons per shaft per day only.

"There is an annual variation in amount of water yield due to alternation of rainy and dry seasons, and cases where decidedly larger flows exist are of short life, the amounts soon dropping away to parity with other shafts in the range."

The bulk of this water is surface water, which can be collected at shallow depths and pumped to surface.

At No. 1 shaft, Robinson Deep, out of 1500 gallons per hour, about one-half is pumped from the 200-foot station, and

in No. 2 shaft, out of a similar amount, 1200 gallons are drawn up from the 250-foot station.

It may be mentioned, as showing the freedom from water sometimes experienced, that these shafts were sunk to their entire depth, No. 1 to 2391 feet, and No. 2 to 1877 feet, without interference from water; when sinking was started, one small steam pump was arranged at each shaft, and this was used for the surface water; below this, all the water was taken up in the skips.

Where water occurs in depth, it generally comes along a dyke or fault. In No. 2 shaft, Rose Deep, there was, along a dyke, an influx of water during sinking which was at times as much as 10,000 gallons per hour, but after the first outburst this gradually decreased.

In Nos. 2 and 3 shafts, Nourse Deep, a considerable amount of water was met in depth, with the occurrence of disturbed ground.

All these large flows of water gradually decrease, after their first outburst, to quantities which are easily dealt with.

The ordinary mine water, as it is pumped from the mine, is generally very impure, as is shown by the following analysis of water from the Spes Bona :—

	Per gallon.
Persulphate of iron	13.40 grains
Sulphate of calcium	6.00 „
Carbonate of „	14.00 „
Matter in suspension	2.87 „
Total sulphuric acid	11.00 „
Total solids	34.18 „

Reaction of the water, acid.

One gallon is approximately 58,485 grains.

Most of these impurities are due to the oxidation of the ore, as is shown by a comparison of the above analysis with one of water taken from the sump of No. 1 shaft, Robinson Deep, just about the time the reef was struck :—

	Per gallon.
Permanent hardness	Nil
Temporary hardness	3.23 grains
Sulphuric acid	Trace
Chlorine	0.64 „
Total solids	19.72 „

Reaction of water, alkaline.

The amount of water met with in sinking the vertical shafts of the deep levels is generally small, causing no serious interference with the operations.

It is generally dealt with in the following manner: the surface water is collected in a sump, at a depth of from 150 to 250 feet below surface, and pumped to the surface by itself by a steam pump generally of the duplex type, about 4 or 6 inches in diameter. This water is collected by a ring cut around the shaft, as described on p. 188, and guided into a chamber which is generally cut into the pump end of the shaft. The water, which comes into the shaft below this, is collected by "ringing" the shaft at the desired points and guiding the water into wooden cisterns placed in chambers cut into the side of the shaft. From these cisterns it is delivered by hose into the skips and taken to the surface after the rock has been hoisted.

Where the water is heavy in a shaft during sinking, it is necessary to put in proper pumps; thus in the No. 3 shaft Nourse Deep a complete 8-inch Cornish pump was put in, because at the 800-foot level, in starting a cross-cut, the shaft became filled to a great height with water.

In order to cope with the larger quantities which follow upon development from these shafts, a complete pumping system is arranged, by which the water is brought to the surface in regular stages or lifts. The intervals between these lifts depend upon the power of the pump which is employed. In the deeper levels it has been arranged to use electrical pumps capable of lifting the water under a head of 500 feet, and the pumps will accordingly be placed and the

sumps cut at this distance apart.[1] The pumps require a chamber about 8 feet by 8 feet by 9 feet, and the sumps have a capacity which depends upon the rapidity with which the water fills into them; they should be capable of holding about two or three hours' supply.

With Cornish pumps the lifts are not placed at such long intervals, 300 feet being a more common distance. In the Nourse Deep No. 2 shaft, where 8-inch pumps of this type are used, there are plunger pumps successively at depths of 320 feet, 660 feet, 1008 feet, and 1356 feet, and another lift will be placed lower down. At the bottom of the vertical shaft, a depth of 1580 feet, a larger chamber has been made for a sump. This sump is to receive the water from the incline shaft, where a separate service will be arranged, so that it requires to be much larger than the intermediate sumps, which only collect the water which falls from above them. At depths of 545 feet and 1222 feet in this shaft, balance bobs have been placed; wherever possible these pump stations are arranged to be at levels from which development proceeds, in order that they may be the more accessible.

With two or more shafts, one is generally made the main pumping shaft, and to this the water from the others is drained; for this purpose the deepest shaft is always chosen.

With incline shafts the same system is used; pump stations are set out at intervals which suit the power of the pump, and they are so arranged that if possible each one shall be placed on a level with one or other of the landing stages, or at some other place which is easy of access. The sumps are generally cut out of the pump end of the shaft, or sometimes they run out into the roof; one of ordinary capacity will hold 10,000 gallons, or on an average about six hours' supply. In outcrop mines the surface water is rendered impure by contact with the oxidised ore, so that

[1] In some of the deep levels Riedler differential air pumps with 1200 feet lifts will be used.

there is no advantage in pumping it separately; in consequence it is allowed to descend to the first pumping station, which becomes the main station, and which is generally placed near the 5th level, about 600 feet along the incline from the surface. This station is usually equipped with larger pumps than the others; thus at the Durban-Roodepoort, from the 5th level to surface, there is a 10-inch Cornish pump, from the 8th to the 5th level an 8-inch one, and from the bottom up to the 8th level a smaller steam-pump.

In the City and Suburban Main Incline shaft there is an electric three-throw pump, with cylinders 6 inches in diameter and 12 inches stroke, on the 12th level at a depth of 1750 feet on the incline, by which the water is pumped to a sump on the 6th level, a height of 850 feet on the incline; from there it is raised by similar pumps, but of 7-inch diameter, to the top of the head-gear at the mouth of the shaft.

Cornish Pumps.—Cornish pumps are extensively used, especially by the outcrop companies in their incline shafts. Owing to the small amount of water which is generally met with in the bottom of a shaft, it is rarely that the Cornish pitwork is taken down there; it is usually arranged that the water from the bottom is pumped up by a small steam-pump into a cistern, from which it is taken by the Cornish pump. Usually these pumps are all force-pumps; if a bucket is put in for the bottom lift, it is generally of the type known as the jack-head pump, in which the bucket rod passes into the working barrel, during the down-stroke, through a stuffing box, and, at the point just above the bucket when at the top of the up-stroke, the water is transferred through an "H"-piece to the rising column, which is not in line with the working barrel. In an ordinary bucket pump the rod is in the rising column for the whole length of the lift. The latter type is often spoken of as working with wet spears, in contradiction to the former, which works with dry spears.

A jack-head pump arranged as the bottom lift is shown

fixed in an incline shaft in Fig. 125, and in Figs. 126, 127, and 128 a plunger pump is also shown fixed in an incline shaft.

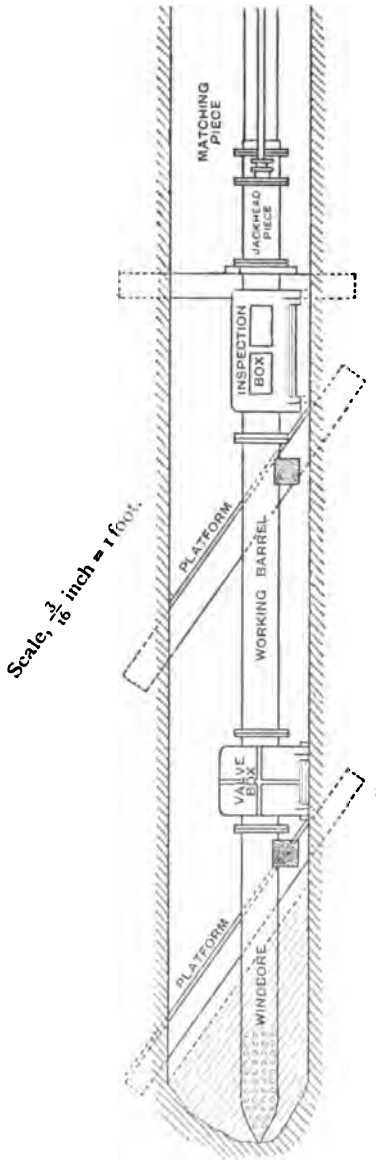
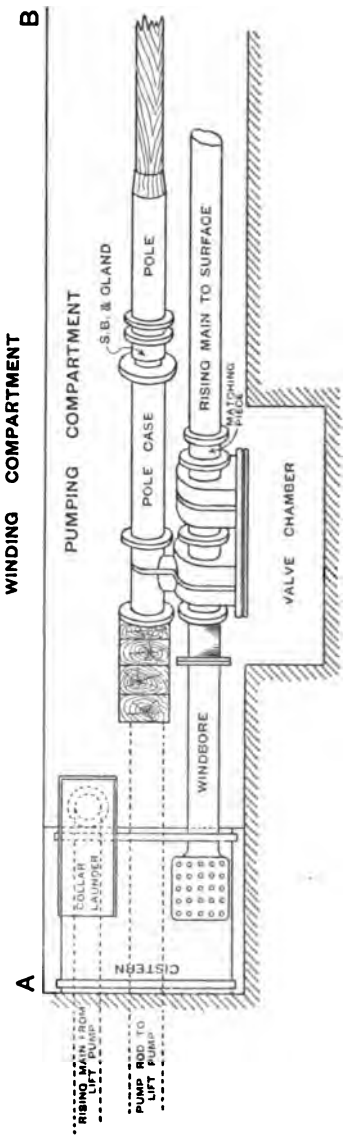


FIG. 125.—Bottom lift pump for an incline shaft.



Scale, $\frac{3}{16}$ inch = 1 foot.

FIG. 126.—Plan of pumping station for an incline shaft. The elevation of this station constitutes Fig. 127.

In these figures the delivery from the lift pump and the arrangement of the cistern and chamber are also shown. The Cornish pumps are worked generally by horizontal, single-

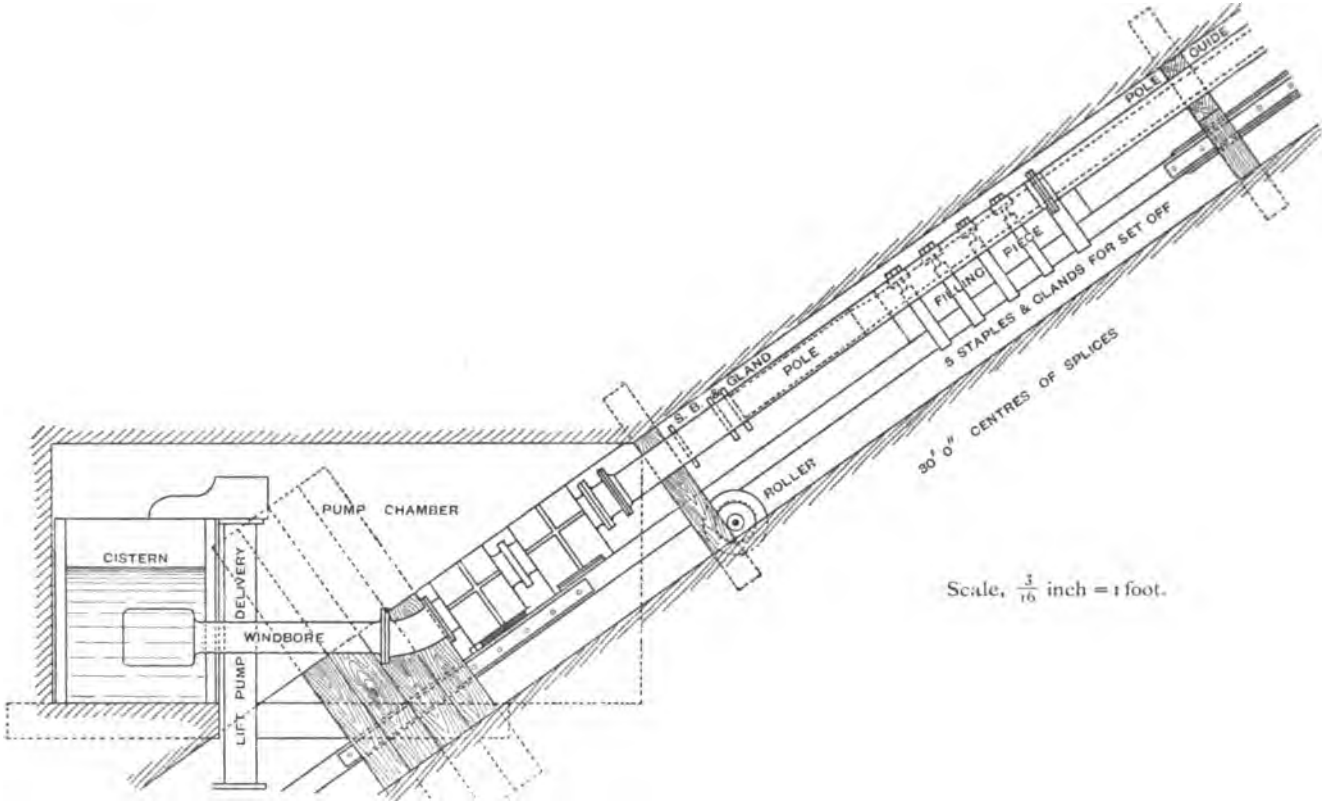


FIG. 127.—Pumping station in an incline shaft. The plan of this station constitutes Fig. 126.

cylinder, geared engines, the usual size of the cylinder being 15 inches diameter and 39 inches stroke, though in some of the deeper mines compound condensing engines are being employed. Pumping engines are fitted with heavy fly-wheels to smooth the irregular nature of the pumping, and also with governors, which will tend to arrest the engine should the rods at any time break.

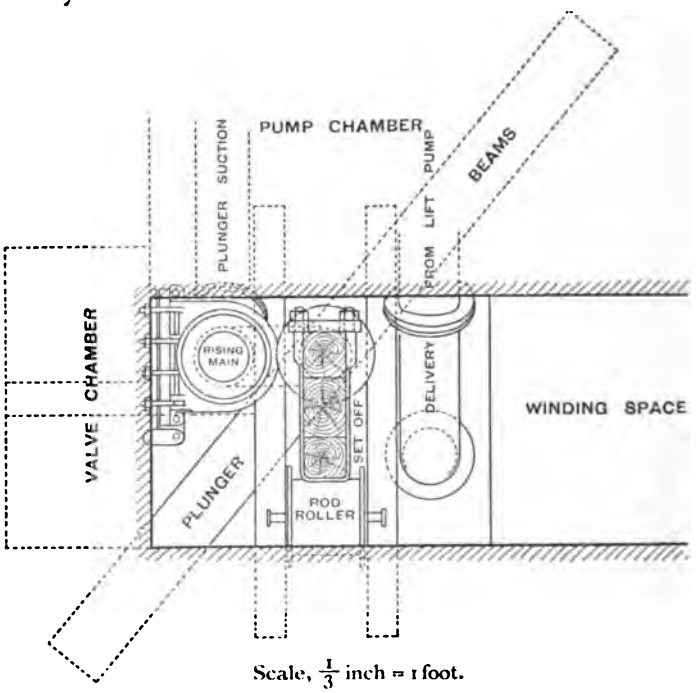


FIG. 128.—Plan of the pump chamber of an incline shaft projected on to a plane at right angles to the plane of the shaft.

The engines are connected with the pump rods by sweep rods and an angle bob, and it is usually arranged that from different points on the engine spur-wheel or disc different lengths of stroke may be obtained, the usual length being about 6 feet.

As the engine-house is not often so close to the shaft that one sweep rod is sufficient, several of them are used; of these the two end ones, one connecting with the engine and the other with the angle bob at the mouth of the shaft, have

a circular movement at one end and a horizontal movement at the other, so that they require to be somewhat stronger than the intermediate ones, which have only a horizontal movement, along which they are guided at all points. These latter are also further supported on wheels, which are fixed to them and which run to and fro on rails in response to the reciprocating movement. Sweep rods are shown in Fig. 129.

At the mouth of the shaft the horizontal movement of the sweep rods is turned into the plane of the shaft by an "angle bob"; it is also usual at this place to counterbalance the weight of the pump rods by a counterpoise, fixed at the end of a horizontal arm added to the angle bob, as in Fig. 129, which then is spoken of as a "balance bob." Where the shaft is vertical the three arms of the bob resemble an inverted T, so that such a bob is termed a "T"-bob.

The pump rods or spears are attached to the angle bob, without any arrangement to take up the curved motion, for by the time the pump barrel is reached this motion practically does not exist. They are usually made of pitch pine, 8 inches by 8 inches in section, and in lengths of about 30 feet; these are butted together and joined by strapping plates of wrought-iron, as shown in Fig. 130.

In vertical shafts the rods hang, so that, if unbalanced, the whole weight is taken by the pin of the angle bob at the top. In order to avoid this, and at the same time to render the work of pumping smoother, balance bobs are put in at regular intervals, about every 600 or 700 feet, to take the weight off the rods and to transfer it to the securely bedded fulcrum of the bob. These balance bobs are in all respects similar to that on the surface, except, perhaps, that they may not be so large. At short intervals down the shaft the rods are guided between cross-pieces. At these places they are protected from wear, by rubbing-boards, which are fixed to them with yokes, so that, when worn, they can easily be

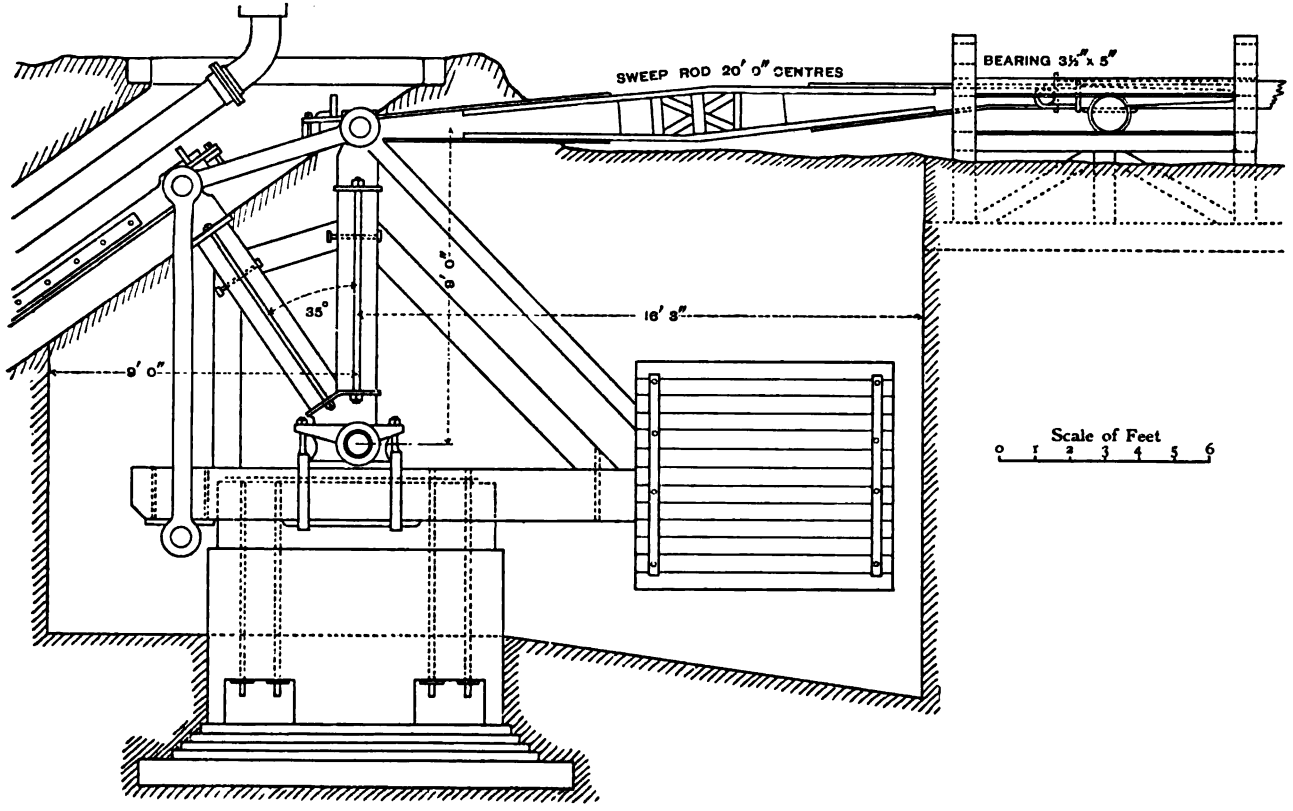


FIG. 129.—Balance bob and sweep rods at the mouth of an incline shaft.

removed and replaced. Catches are put in occasionally to arrest the rods should they break; each catch is made of two strong bearers placed at such a distance apart that the rods move between them with not more than the necessary amount of clearance. To the rods shoulder-pieces or "wings" of hard timber are strapped at such a height, that at the bottom of the down-stroke they almost reach the bearers. These shoulder-pieces are of such a size that they could not pass between the bearers, so that should the rods break the greatest possible

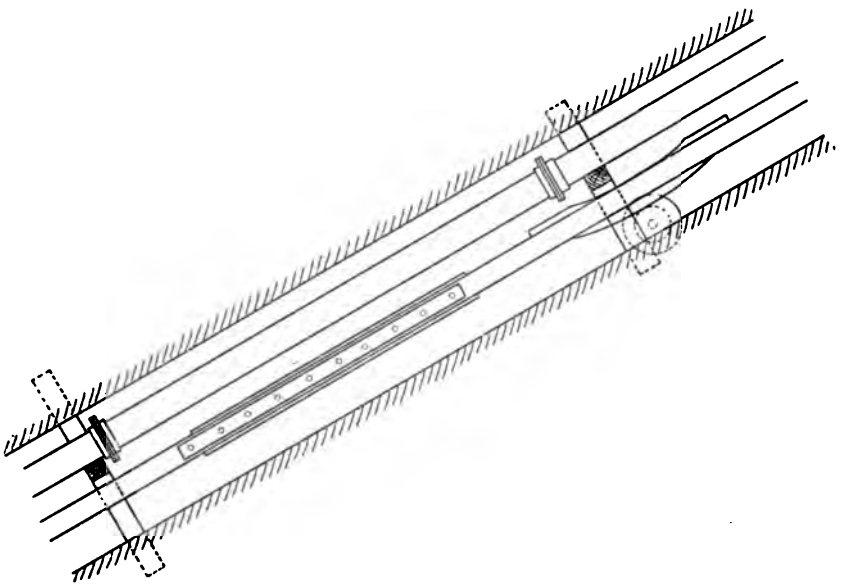


FIG. 130.—Pump spears in incline shaft.

fall would only be a little more than the length of the stroke. In order to prevent splintering, these catches and the shoulders where they would come in contact are sometimes covered with iron plates. In incline shafts, where the angle of dip is about 30°, the rods are to a great extent supported and carried on cylindrical cast-iron rollers, which are fixed to posts in the shaft, as shown in Fig. 130, at intervals of about 30 feet; the remaining and smaller portion of their weight is suspended, but as this is not so considerable, balance bobs are not so often required as they are in vertical shafts.

The posts to which the rollers are fixed are also used as guides, between which the rods, protected with rubbing-boards, move. Near the surface the rods are lifted off the first two or three rollers during the greater portion of the stroke, because of the curved movement of the bob; in order to support their weight at these points, pieces of wood are fixed to them underneath, which are of such a curved shape that they keep touch with the roller for the greater portion of the stroke, as shown in Fig. 130.

At the required points the plunger poles are set off from the rods; this is done either by strapping the plunger pole, with staples and glands, to the rods, with a filling piece between the two, as shown in Fig. 127; or by setting off a cast-iron bracket which is directly attached to the rods, and from which the pole proceeds as shown in Fig. 131. In the first method the free end of the pole is often extended backwards and made to pass through a guide, which renders the working more true.

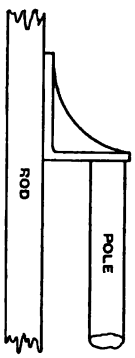


FIG. 131.—Diagram illustrating a method of setting off the plunger pole from the rods.

Where there is a bucket lift at the bottom, the wrought-iron bucket rod is fixed to the end of the main rods, by inserting its top end, which is suitably shaped, into an iron ferule at the bottom of the main rods, where it is fixed by a key.

The pumps themselves are supported in vertical shafts on bearers; for each pump two bearers, about 3 feet deep, are placed across the shaft with their centres about 3 feet apart; on top of these, cross-pieces of hard wood and of large section are placed to form a platform for the bottom of the "H"-piece. In order that the pump may run truly, these bearers and cross-pieces are put in perfectly true by means of a spirit level. Where possible, it is better to obtain a shoulder of solid rock in the place of one of these bearers.

In incline shafts the pumps are similarly supported, though in Fig. 126 the "H"-piece is shown as supported only under the line of the plunger.

In vertical shafts the column, though supported principally at the pump-bearers, is also supported at intervals; in these cases the flanges are shouldered against pieces carried by the shaft timbering. In incline shafts it is usual to place the columns on the floor of the shaft, though they are sometimes supported in a position up near the roof, on horizontal pieces placed on the guide pieces. The column is further supported by resting its flanges in places against shoulders formed by the shaft timbering, and sometimes it is held by chains in order to prevent it from slipping down the shaft.

A column 10 inches in diameter is often used in the main pumping shafts. To deal with less water, or in order to supplement the main pumps, smaller columns are used.

In the main shaft of the Village Main Reef, in consequence of a great influx of water which occurred during sinking, a rising main or column 16 inches in diameter was put in. This influx has now, however, in great part subsided.

A 10-inch pump requires about 12 H.P. per 100 feet of lift to run it at the ordinary rate, when it is capable of raising 15,000 gallons of water per hour. There are very few mines where there is such an amount, so that a 6-inch pump will take all the water in most cases, but a larger pump is a wise provision against flooding. Such a pump can, on ordinary occasions, be run at a slow speed, or it can at any time, after having pumped out the water, be stopped for repairs if necessary whilst more water is collecting.

Steam Pumps.—As stated before, for sinking shafts or winzes, where only the usual small amount of water is met with, small steam pumps are used to throw the water up to the main pumping stations. The ordinary type used is the bucket type, of which the steam cylinders are larger than the water cylinders, and these again larger than the delivery

pipes. A pump with a 3-inch delivery pipe will have a water cylinder about 4 inches in diameter, and a steam cylinder about 6 inches. Such a pump at an ordinary working pressure is capable of throwing 3000 gallons per hour under a head of 180 feet. These pumps can be worked with compressed air. It is rarely that they are used to throw the water to surface.

Force Pumps.—The type of pump which has lately been adopted to a considerable extent is the electrically driven two-throw or three-throw plunger pump.

The three-throw pump has three cranks, arranged at angles of 120° to one another, on the pump shaft. From each of these cranks a plunger is worked to and fro in its water cylinder, which is usually about 5 inches diameter and 10 inches stroke.

The motor shaft is run at a high speed, and from it the pump shaft is driven by gearing, so that the speed is reduced to about 40 to 60 revolutions per minute. These pumps have been placed in some of the deep shafts to throw water under heads of 500 feet. The two-throw pump has only two cranks, otherwise it is identical to the other. Of the two the three-throw pump is better, because with three cranks the strain is more equally distributed, and the pump works more smoothly.

Costs of Pumping.—The costs of pumping are generally included with those of hoisting, and a statement of the combined costs of these two operations is given on page 269. In the case of the Crown Reef the expenses for pumping are given separately, and there it is interesting to note that the cost of maintenance is relatively very high.

CHAPTER XIII

DEVELOPMENT

WITH but one or two exceptions the actual development of the ore is taken to start from the shaft, which it does not include, and to cover all those operations, such as driving, cross-cutting, sinking, rising, which result in opening up the ore, so that it can afterwards be systematically mined. The relative positions of the shafts, drives, winzes, cross-cuts, etc., are shown in Fig. 149.

It is usual to regard the main shafts as permanent works, because they are made to last out the mine, and in this respect they are in the same position as all other permanent works, such as dams, reservoirs, etc., and as all plant and machinery. Development is different because the drives, cross-cuts, winzes, etc., are only in use as long as there is ore standing above them, and they are, in this sense, not permanent.

In the Crown Reef, development is taken to include only the main shafts and main cross-cuts, the drives, etc., which are ordinarily looked upon as constituting development, being included with mining proper.

In those deep levels which are controlled by the Rand Mines, Limited, it is usual to regard the incline shafts as part of development, and the vertical shafts as being permanent works. Before these mines start crushing, development is charged to capital account, whilst after crushing has commenced it is charged to working expenses.

The state of the development of a mine is expressed by the number of tons of ore which are blocked out ready for mining, *i.e.* stoping. It is best appreciated in the deep-level mines, where the work of development, and that of mill erection, are always so planned that by the time the mill is running, a large amount of ore is opened up ready to be broken.

Drives.—The drives extend horizontally on the reef plane, in direction parallel to the strike. They are usually about 6½ feet high and 6 feet wide at their widest for a single track. A double-track level, such as is driven in the deep levels, is generally from 9 to 10 feet wide, and from 7 to 8 feet high. The various sizes of the development headings in the Jumpers Deep are given on p. 385. The ordinary section is only approximately rectangular, it being more usual to make the greatest width about half-way up from the floor. Drives are placed on the reef plane at a depth apart which varies from 100 to 200 feet, and is, on an average, about 140 feet. In the deep levels the larger depth is more frequently found than in the outcrops. It is this depth, from one level up to the next above it, which is spoken of as the "backs" of a level.

Drives are started off from the shaft as the correct depths are reached. In the Robinson the shaft is sunk for five levels at a time, and then sinking is stopped while all the five drives are pushed forward to the boundary before active sinking is again continued; but it is usual to sink the shaft more steadily.

The drives are usually carried on the reef itself, so as to expose it as much as possible in order that some idea of its value may be obtained by sampling. In the Geldenhuis Deep mine it was tried for a time to run the drives in the footwall and independent of the reef, but this method was found not to be advantageous. Where the reef is small and rich, the drive is so carried that the reef is just exposed in the hanging wall corner of the drive, and but little of it is broken; for with a preponderating percentage of waste, the reef which is broken in driving becomes to a great extent worthless.

In order that water and ore may travel easily along the drives it is necessary to give these latter a slight gradient; in the majority of cases the water and the ore travel in the same direction, *i.e.* towards the shaft, so that the gradient given for one is generally useful for the other. It is found by experience that the gradient most suitable for tramming, taking into consideration that the empty trucks have to be pushed back, is one of three-quarters per cent down towards the shaft. This, however, is barely sufficient to allow the water to run easily, unless a small ditch is made on one side and kept very clear, which is not easy where a lot of tramming is going on, so that it is better to give the level a grade of about one per cent. Where, however, a level is driven for a main tramming way, facility for easy tramming is the first consideration. In the Robinson mine the ore from the lower levels of the East Incline will be taken to the West Incline, along the eleventh level, which for this purpose has been carried in a straight line from one incline to the other, with a width of 10 feet and a gradient of 1 : 114 in the direction opposite to that which the loaded trucks take, but as mechanical haulage will be used the direction of gradient is here a secondary consideration.

Very little ore has been won from "adit" levels, because the even nature of the surface does not allow of this cheap method of exploitation. In the earlier days of the fields, some amount of ore was extracted through them, chiefly in the Florida and Roodepoort districts.

Intermediate drives are only made where, generally by reason of dyke intrusion, any piece of ground cannot be included in the regular plan of development of a mine. As a rule, they are only used to open up relatively small areas of ground. They are often seen along the intersection of the reef with a longitudinal dyke, by following which they sometimes rise from one level to another as inclined drives.

In addition to the main drives smaller "stope drives" are

often made about 5 or 6 feet above the main drive, on the plane of the reef, in order to facilitate stoping. Where these are made some time previous to stoping they are considered as belonging to development; but when they are being driven only a few feet in front of the advancing stope face, they are considered to belong to stoping or mining proper.

It is not often that any level continues from one mine into the adjacent mines. In the East Rand Proprietary mines, however, a main drive will pass through them all.

Driving.—Driving is either done by hand or machine drilling. In the soft and unoxidised ore near the surface, hand drilling was used to a great extent, but in the blue ground and in the ordinary quartzite, machine drills are almost exclusively used, the great advantage being the higher rate of development which can be maintained by means of them. By using machine drills, and in the absence of an adequate supply of native labour, the boys, who would otherwise be required for development work, can be employed in drilling in the stopes, where they work more advantageously than elsewhere. These remarks apply to all other headings, especially to raises, where native labour is at its worst in drilling. In sinking winzes they do very well.

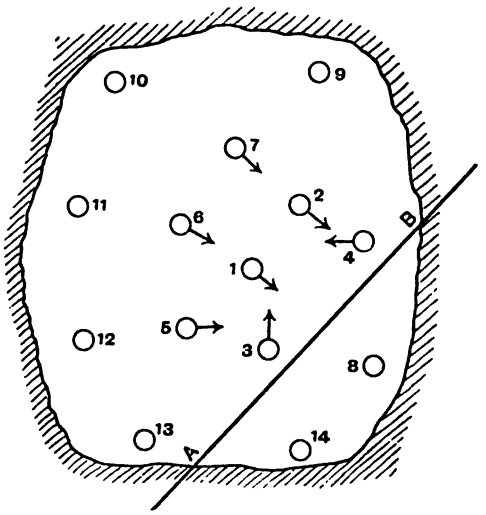


FIG. 132.—Face of a drive drilled over for a side cut.

The ordinary method of advancing the end of a drive is to take out a "cut," and to break the remaining rock into the cut. Where the rock is well bedded, a side cut is made to break against a plane well down on the footwall side of the drive. In Fig. 132 the disposition of the holes over the face

to effect such a cut, and to square up afterwards, is shown. Nos. 1, 2, 3, and 4 are the cut holes proper, and by blasting these together the cut is brought away from the bedding plane A, B. After this 5, 6, and 7 are blasted to enlarge the cut, and then the outside holes. Of these last, those in the floor, called "lifters," are charged with such a length of fuse that they go off last and loosen the broken rock which has already fallen, so that it is more easily shovelled.

It is, however, more usual to take the cut out from the

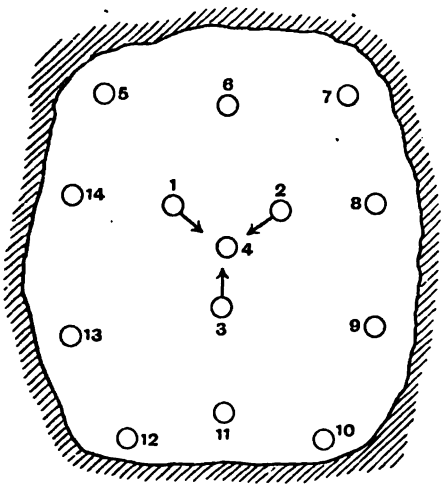


FIG. 133.—Face of a drive drilled over where the rock is of average hardness, centre cut.

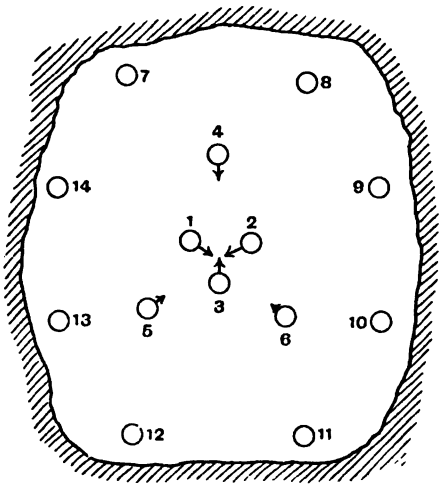


FIG. 134.—Another method of drilling over the face with rock of ordinary character.

centre, the number and arrangement of the holes depending upon the character of the rock. In Fig. 133 the arrangement in ordinary ground is shown. Nos. 1, 2, and 3 are the cut holes proper. They are about 6 feet long, and inclined towards each other, so that they often meet. No. 4 is a short hole, about 2 feet deep, placed to break the collar of the cut. These four are blasted together, the fuses being so arranged that No. 4 goes off first. The top holes, 5, 6, and 7, are as nearly horizontal as possible, though they are generally found to look up and outwards a bit. Nos. 6 and 11 form "easers" for the top holes and lifters respectively.

In Fig. 134, with the same number of holes, a different arrangement is shown. The three cut holes, 1, 2, and 3, are placed very close to one another, the distance between any two being about a foot, and there are three easers, 4, 5, and 6, to enlarge the cut.

For easy ground the arrangement shown in Fig. 135 is used. The four cut holes are placed at the corners of a square, and so drilled that they converge, each one towards that other which is on the same horizontal plane. Altogether twelve

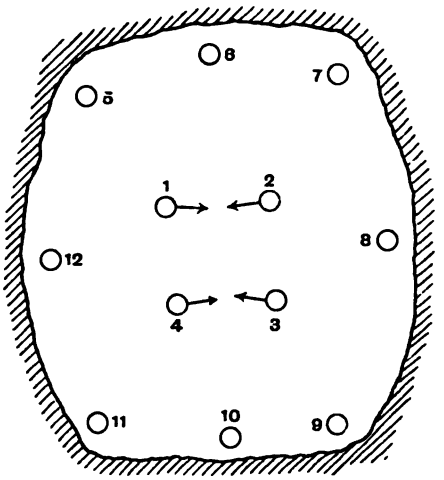


FIG. 135.—Arrangement of holes in the face of a drive for easy ground.

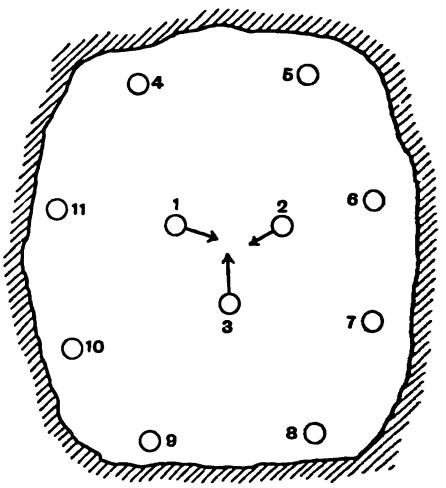


FIG. 136.—Another arrangement of holes in the face of a drive in easy ground.

holes are required. Another arrangement for easy ground or for a smaller drive is shown in Fig. 136. Where the ground is very tight more holes are required. The arrangement shown in Fig. 137 is for a face where the ground is very tight. The cut holes, 1, 2, 3, and 4, are about 5½ feet long, and they each take six sticks of gelatine. No. 18 is a smaller hole, about 4 feet deep, which is charged with about five sticks. Nos. 5 and 6 are sometimes called shoulder holes; 7, 8, and 9 are three easers; 10 and 11 are the top or back holes; 12, 13, 16, and 17 are side holes or skimmers; 14 and 15 are lifters, and these are fired last. To square up a face of this sort, with a cut

of 5 feet, requires about six packets of gelatine, or about 60 lbs. weight; where the ground is very tight as much as 75 lbs. is sometimes required. With a drive of average rock about one case, containing 50 lbs. nett, of gelatine is used. Allowing the average sectional area of the drive to be 36 square feet, and the length of cut to be $4\frac{1}{2}$ feet, it works out that with machine drills each ton of rock blasted out from the face of a drive requires about $3\frac{1}{2}$ lbs. of gelatine, which is higher than in shaft sinking, and very considerably higher than in stoping.

On the eighth level in the Durban-Roodepoort, during one month, one end was driven 225 feet. The drive was small, about $6 \times 5\frac{1}{2}$ feet, and eleven holes covered the face. During the day there were two shifts, and during each shift a cut of from $3\frac{1}{2}$ to 4 feet was taken out and squared up.

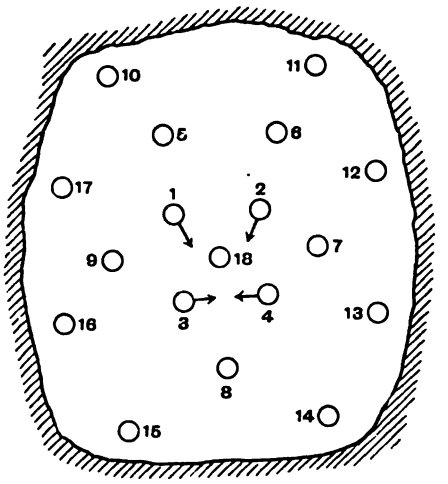


FIG. 137.—Arrangement of holes in the face of a drive where the ground is tight.

The rate of progress in driving, where machine drills are regularly employed, is often well over 100 feet per month. In the Ferreira during the year ending 31st December 1896 the average amount driven per month by each rock drill in the drives was 107.3 feet.

When any drive is being pushed it is usual to work in three shifts of eight hours each. At an ordinary rate the day is divided into two shifts, of which, when the driving need but proceed leisurely, one alone is worked, the drill-man choosing that one which, from the smaller number of machines running, will allow him the more air.

It is usual to run the drives either on contract or with a bonus, the men, as a rule, preferring the latter. The bonus is given so much for every foot which is above and beyond a

footage determined on by the mine foreman as being good work. Thus in the New Primrose the men in the ends get bonuses after 18 or 20 feet per week have been driven, two shifts being used. The North Reef (Main Reef proper) and Main Reef (South Reef proper) are in tight ground, and on them the bonus is given after the smaller number of feet; whereas on the Middle Reef (Main Reef Leader), a bonus is only paid after 20 feet per week have been driven, because the ground is easier, owing to the presence of a clay seam. The contract prices or bonus allowances vary according to conditions. Sometimes the drill-men are required to shift their own dirt, but generally they only have to remove it far enough from the face to enable them to rig up their machines again. In other cases special boys working for the company proceed to remove the dirt from the face almost immediately after blasting. Along the Rand the character of the ground does not vary much; in the Roodepoort district, however, it appears to be easier than elsewhere. The contract price paid to a contractor, who has to provide labour, explosives, lights, lubricants, but neither machines nor air, and who has to shift his dirt back from the face, but not to tram it, is generally from 35s. to 47s. 6d. per foot; that for stope drives is but little less.

Winzes, Sinking and Raising.—Winzes are headings which are driven on the reef plane, at right angles to the drives, and they make connection between the drives on the different levels, as shown in Fig. 149. They run parallel with the shafts, but, unlike the shafts, they are usually only continuous between two levels. They are either made by sinking or by raising. In the latter case the heading is spoken of as a "raise," until connection is made through to the level above, when it is called a winze. They are placed at a distance apart, which varies from 200 to 500 feet. Winzes are looked upon as lines from which stoping can be commenced; the distances between them along the drives depend upon the amount of driving which has been done, because the

amount of ore required for the mill being the same in any case, the number of lines from which it can be broken must also be the same, so that where the drives are well ahead, these winzes or lines can be placed farther apart that when the drives are backward. In the deep levels, where the development from the start is kept well ahead, the winzes are from 400 to 500 feet apart. Up to a limit fixed by the loss of interest on the money spent in advance development, the smaller the number of winzes, *i.e.* the greater the distances between winzes, the less the cost of development.

Though between the various levels the winzes are not in continuous line, it is usual to find that good average lines run down through the mine at the regular distances apart, and all those winzes which take part in forming any one average line are named under one heading, thus a winze called "No. 1, West of the Main Incline," would be taken to cover that line of winzes nearest to, and west of, the Main Incline shaft. To specify any particular winze the level from which it was sunk is mentioned.

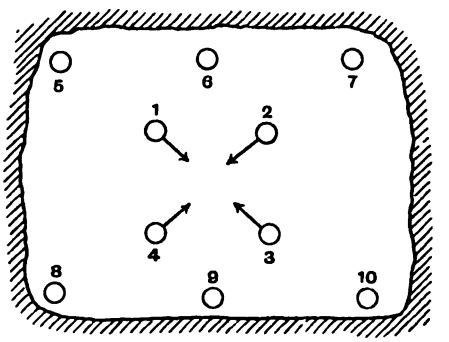


FIG. 138.—Usual system of drilling over face of winze.

The usual size for a winze is about $4\frac{1}{2}$ feet in height and 6 feet in width; this is generally blasted out by ten holes drilled as shown in Fig. 138; 1, 2, 3, and 4 are cut holes; 5, 6, and 7 back holes; 8, 9, and 10 are lifters.

In most mines winzes are made chiefly by sinking; in the Crown Reef they are often right down to the next level, before the drive has advanced to meet them, whereas in the Robinson they are made almost entirely by raising, it being usual in that mine for the man who is driving the levels with machines to come back and put up the raises. The native is better in

sinking than in advancing any other heading, so that in many cases the winzes are put down by hand labour to connect with raises which are being put up with machine drills from the level below. With machine drills it costs slightly more to sink a winze than to raise, and usually it again costs slightly more to raise a heading 6 feet by $4\frac{1}{2}$ feet than to drive one 6 feet by $6\frac{1}{2}$ feet horizontally.

Cross-cuts and Cross-cutting.—Cross-cuts are headings advanced horizontally across the formation, either for tram-

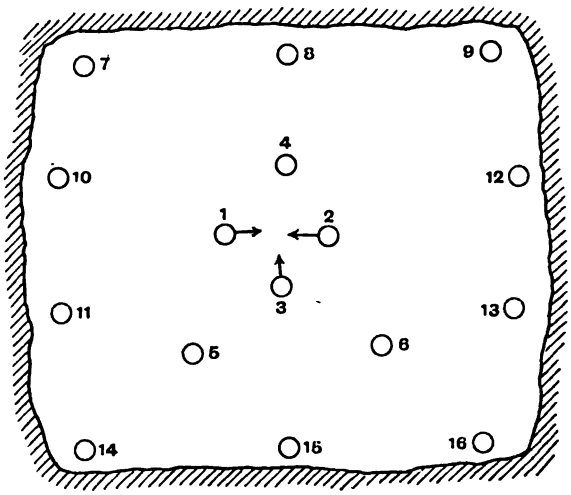


FIG. 139.—Face of a cross-cut drilled over in easy ground.

ming ore, for drainage, or for prospecting purposes. An ordinary cross-cut is of the same size as a drive, but those which are made from the shaft, generally called main cross-cuts, are larger, common measurements being 7 feet in height by 8 feet in width.

There are two ways of advancing the face of a large cross-cut; either a central cut is taken out and the remaining rock blasted into the cut, as in driving; or the top portion is advanced after the cut has been taken out, and afterwards the floor is lifted.

In Fig. 139 the disposition of holes for the first method is shown where the ground is easy; 1, 2, and 3 are the cut holes;

4, 5, and 6 are the easers; 7, 8, and 9 are the back holes; 10, 11, 12, and 13 are side holes; 14, 15, and 16 are the lifters.

In Fig. 140 a face, 7 feet high by 9 feet wide, is shown drilled over to carry out the second method; 1, 2, and 3 are the cut holes; they are blasted first, and then numbers 4 to 12, leaving the floor to be afterwards lifted by numbers 13 to 15. It is usual in cross-cuts of this size to drill with two machines rigged up on separate bars placed upright, though sometimes the two machines are clamped on to the same horizontal bar.

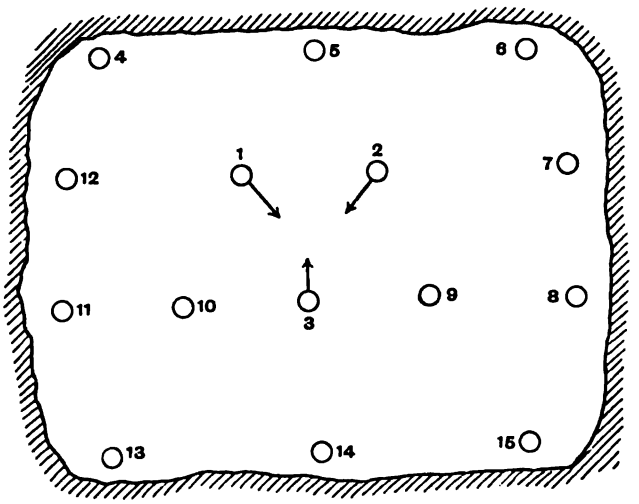


FIG. 140.—Face of a cross-cut drilled over to take the top portion first and to leave the floor to be lifted separately afterwards.

In consequence of the larger number of holes drilled, the greater amount of gelatine used, and the larger amount of dirt to be shifted, these large cross-cuts cost more than ordinary drives. One of the longest along the Witwatersrand is that on the 4th level No. 2 shaft, George Goch, which is 740 feet long; it is for a double track, and cost, fully equipped, £12 per foot. Where a cross-cut is being driven in a direction towards the dip, the rock breaks in such a manner from the bedding planes that it is not possible to place the holes so advantageously or with a proper burden upon them, as where the cross-cut is going in the opposite direction; in

consequence along the Rand it is usual to pay more for one being driven to the south, than for one to the north.

In the Ferreira the main cross-cuts are 7 feet high and 12 feet wide. During 1895 two drills used in advancing such a face accomplished on an average 110.6 feet per month.

Costs of Development.—The different conceptions of what is included in development render it very difficult to get at the real cost of this item. The following figures of costs are based on statements in the annual Reports of the Companies mentioned. In the George Goch, during the year ending 10th October 1896, the total footage driven, risen and sunk, was 11,375 feet, at a cost of £4 : 7 : 0¾ per foot, but this included sinking the three main shafts an aggregate depth of 750 feet and advancing 970 feet of double-track main cross-cuts. If it be estimated that the cost of these be £12 per foot, and if they be taken out of the calculation, there remains 9655 feet of ordinary development which cost approximately at the rate of £3 per foot. Of the total footage, 6873 feet were driven by compressed-air drills, which gave a footage per drill per month of 81.8 feet.

In the Crown Reef, during the year ending 31st March 1897, drifting, rising, and sinking, which ordinarily constitute development, cost for a footage of 10,232 feet at the rate of £3 : 3 : 6.592 per foot, whilst the shafts and cross-cuts, which in this mine constitute development, cost approximately at the rate of £11 : 9s. per foot for a total of 828 feet.

In the Henry Nourse, during the year ending 30th June 1897, 235 feet of main shafts were sunk at a total cost of £2319 : 14 : 5, or roughly, £9 : 15s. per foot, and 8650 feet of development, made up of drifts 6580 feet, winzes and raises 457 feet, and cross-cuts 1613 feet, cost £36,083 : 15 : 3, or about £4 : 3s. per foot.

In the Meyer and Charlton, from 1st June 1896 to 31st December 1896, the following footage was accomplished at the following cost :—

	Feet.	Inches.		Total Costs.		
Drives	2081	0	European wages	£1,640	0	0
Cross-cuts	326	6	Natives, including food	1,807	7	8
Winzes	630	0	Explosives	795	16	2
Raises	721	0	Tools, stores, fuel, etc.	2,310	12	7
			Smithy account	631	5	5
Total	3758	6	Contractors	4,796	12	1
				£11,981	13	11

The cost per foot works out from these figures at about £3 : 3s.

In the Robinson, during the year ending 31st December 1896, the following total footage was accomplished :—

Main shaft sunk		655 feet
Drifting	6,267 feet	
Raises	3,384 „	
Cross-cuts	1,542 „	
Total Development	11,193 feet	

It is interesting to note from these last figures that the winzes are in that mine made by raising and not by sinking. The accompanying statement, which, in addition to the above figures, is taken from the General Manager's Report, shows the details of the cost of the latter total footage of the 11,193 feet of drives, raises, and cross-cuts :—

	£	s.	d.	Total Costs. £	s.	d.	Cost per Foot. £	s.	d.
Superintendence—									
Proportion of wages of mine captain, shaft bosses, rock-drill foremen, etc.				937	10	0	0	1	8.10
Day Work—									
Pipe fitters, platelayers, timbermen and miners	7255	13	0						
Natives and food	1826	1	3						
Explosives	9132	10	6						
Timber, tools, oil, candles, etc.	555	18	0						
Rails, trucks, etc.	1091	8	10						
				19,861	11	7	1	15	5.87
Carry forward				20,799	1	7	1	17	1.97

	£	s.	d.	Total Costs. £	s.	d.	Cost per Foot. £	s.	d.
Brought forward				20,799	1	7	1	17	1.97
Tramming—									
Muckers and Tramming .	245	17	0						
Natives and food . . .	1231	19	0						
				1,477	16	0	0	2	7.70
Hoisting—									
Engineers, banksmen and bell-tenders	704	13	6						
Natives and food . .	119	11	0						
Engine and banksmen's stores	142	9	5						
Fuel	813	10	4						
				1,780	4	3	0	3	2.17
Compressed Air—									
Proportion of compressor costs				4,366	17	9	0	7	9.63
Proportion of cost of rock-drill maintenance . . .				1,115	7	3	0	1	11.91
Tool Sharpening—									
Blacksmiths . . .	1089	7	6						
Helpers and food . .	372	13	6						
Smiths' coal . . .	309	19	0						
Proportion of workshop expenses	76	19	0						
				1,848	19	0	0	3	3.65
Rock-drill Steel . .				349	19	9	0	0	7.50
				£31,738	5	7	£2	16	8.53

In the Simmer and Jack, during the ten months ending 31st December 1896, the development consisted of:—

Drifts	17002.5 feet
Winzes	2705.5 „
Raises	3402.5 „
Cross-cuts	2597.5 „
	25708.0 feet

This was charged as under:—

Excess development . . .	£100,173	2	4
Development	26,170	10	4
	£126,343	12	8

or at the rate of £4 : 18s. per foot.

In the Geldenhuis Estate, during the year ending 31st March 1897, the rock cut was as under :—

	Feet.	Inches.
Driving	4295	3
Rising winzes	449	0
Sinking winzes	1063	9
Sinking shafts	448	9
Cross-cuts	839	3
	7096	0

This was done at a cost of £25,737 : 7 : 2, or at the rate of about £3 : 11s. per foot.

The following figures of the Ferreira are taken from the General Manager's Report for the year ending 31st December 1896 :—

Rock cut.

Shaft-sinking		449½ feet
Drifting	3651 feet	
Winzes and rises	1565½ "	
Cross-cuts	740½ "	5957 "
		6406½ feet

The development expenditure during 1896 was £27,691 : 3 : 5, so that the cost per foot works out at about £4 : 12s. per foot.

The following table shows the distribution of the rock drills employed at this mine during the year :—

Shaft-sinking	0.24
Driving	3.03
Rising	0.64
Rising for ore bins and shoots	0.30
Cutting shaft stations	0.08
Stoping	1.91
The average number employed being	6.2

The average amount of work performed per rock drill per month during the year was :—

Sinking	54.6 feet
Driving	107.3 „
Rising	71.3 „
Rising for ore bins and shoots	91.4 „
Stoping	50.4 cubic fathoms

It is interesting to note from the distribution of the rock drills that none were employed in sinking winzes; also that greater speed is obtained in driving than in rising.

The following is a statement of the cost per foot of development at the Princess Estate during March 1897, which was kindly supplied me by Mr. White, the manager:—

	Cost per Foot.		
Cash paid contractors for personal wages	£0	13	6
Explosives	0	13	6
Compressed air	0	7	0
Hoisting rock broken during development	0	3	0
Wages paid native machine helpers	0	3	0
Tramming rock broken during development to shafts	0	2	3
Management	0	2	0
Maintenance of machine drill	0	2	0
Air piping	0	1	9
Track-laying	0	1	6
Pumping	0	1	6
Drill-sharpening	0	1	9
Steel	0	1	6
Candles	0	0	3
Food for native machine helpers	0	0	6
	£2	15	0

From the cost of explosives per foot given above, it may be calculated that, with a drive 6 feet high by 5½ feet wide, such as is used at the Princess, the amount of explosives consumed per ton of rock broken from the face is just over 2 lbs. weight.

The costs of development per ton of ore milled are given in Chapter XX.

Any body of ore may be considered to be developed when, adhering to the regular system of development, a drive has been advanced to a position immediately underneath it; the

winzes, if only for purposes of ventilation, follow closely upon the advance of a drive.

The calculation of the amount developed is made by multiplying the area on the reef plane by the thickness of the reef, the term "reef," in this connection, being often taken to cover both the banket and that thickness of interstratified waste which must necessarily be mined with it, though when the reef is small or the thickness of such waste is large, then the amount of ore developed is calculated on the "milling" thickness of the reef, which is the thickness mined less the amount sorted out. The number of cubic feet of solid ore is thus obtained. As the result of some experiments by Mr. Franklin White, it is substantiated that about 12 cubic feet of solid banket weigh 1 ton of 2000 lbs., so that by dividing the number of cubic feet so obtained by 12, the tonnage of ore developed is obtained.[1]

In measuring up ground which is fully developed, the area on the reef plane which is occupied by dyke intrusions is subtracted from the total area. In addition, a small allowance has to be made for those pillars which will never be taken out. The proportion of ore left in as pillars varies from $2\frac{1}{2}$ to 10 per cent, depending upon the inclination of the reef and upon the nature of the roof. It is likely that in the deep levels the lower figure, $2\frac{1}{2}$ per cent, will not be exceeded.

A rough way of estimating the amount of ore developed is to allow so many tons for every foot of drive driven on the reef. This number of tons is obtained from the length of backs and the thickness of the reef, and it is a constant multiple for each individual level. Thus in the City and Suburban it is reckoned that every foot so driven develops 20 tons of ore; in the Wemmer about 30 tons.

In estimating the number of tons of ore in any undeveloped area, the same factors are used, and from the result obtained

[1] A table of tonnages per claim with varying dips is given in Appendix I., p. 484.

an intelligent allowance is made for the loss which is sure to be caused by dyke intrusion. In the absence of any correct figures, it is usual to deduct 10 per cent for dykes and faults. It is probable that this is in most cases a very liberal amount. Some allowance must also be made for the boundary pillars which the law requires to be left, and for the pillars to support the roof.

Development costs are a part of the mining costs, so that they have to be calculated on the ton as the unit. There are two principal ways of doing this; either all the costs of development are charged to a capital account called "mine development," to be redeemed by a charge on the ore as it is mined, or they are charged at once as part of the mining costs of the ore which is being mined at the same time, and which has been developed previously.

The first method was very well described by Mr. Lionel Phillips in his speech at the general meeting of the Ferreira, 24th November 1891, in the following words:—

"The opening of a level was charged to development, that was to say, the sinking of winzes down to the level and driving the tunnel on the level went to the debit of development and formed the cost of developing that level.

"The Board then considered the best means of redeeming that amount as the extraction of the ore took place, and therefore opened the 'development redemption' account. The width of the reef in the winzes and on the level formed the basis of calculations for the number of tons which the level might be expected to yield. That tonnage was divided into the cost of development, and by that means the amount charged per ton as the ore was extracted was arrived at. That was placed to the credit of 'development account' and charged to the 'development redemption' account; thus as each level was exhausted, the cost of having developed it had been credited to the 'development account' and charged to the 'development redemption' account.

"The object with which that was done was that in case the Board for any reason should decide to develop at a much greater speed than the rate at which the ore was extracted, there would be an account which would show to its debit the amount of development in excess of extraction; likewise, on the other hand, if the rate of extraction were greater than the rate at which the next level would be developed, the mining expenses would not appear less than they should appear on that account."

This method is the more frequently used, and where the rate of development is liable to differ from the rate of extraction, it is undoubtedly the more accurate. By using it the rate of development is reflected in the balance-sheet, where the value of the exhausted development is periodically stated. Any drop in this asset would indicate that the reserves had been drawn upon, and any rise that, while the mill was kept supplied, the reserves had nevertheless been increased.

In the second method the costs which were incurred in development during the time that a body of ore was being extracted are charged to the mining costs of that body. The accuracy of this method demands that the rates of development and extraction shall be the same. The charges of development then go immediately to working costs, so that there is no possibility of debiting the capital account with items which should properly be charged to working costs.

This system can, however, only be adopted when the mine is fully equipped and running. In the deep levels of the Rand Mines group all underground work, whether in shafts or for development, is charged to capital account until the mill starts, after which all the underground expenses are charged to working costs, and the previous expenses are regarded as though for permanent works, appearing in the balance-sheets as assets, to be written down annually by depreciation.

A third method of charging development is described on p. 440. By this the excess development each month is

charged to capital account, and the ordinary development is placed at once to the working costs. This method is used chiefly in the mines controlled by the East Rand Proprietary Mines, Limited.

When a mine starts developing, the cost per foot driven is great, because the general charges have to be distributed over such a comparatively small amount of work, and because the most expensive work, such as cutting ore bins, pump lodges, stations and main cross-cuts, has then to be done. As the number of headings increases, the cost of development is diminished, but the minimum and normal is not reached till milling commences, when the general charges are distributed over a greater number of accounts. This is illustrated by the following figures from the Jumpers Deep. In July 1896, when development was practically commenced, 499 feet cost £9 : 8 : 9 per foot, and in December of the same year at No. 1 shaft, 1093 feet were extended at an average cost of £5 : 12 : 8 per foot.

In the deep levels before mentioned, the practice of charging all expenditure before the mill starts to capital account, places these high development costs to the same account, from which they are not redeemed, but gradually written down by depreciation. This is the fairest way of treating them, for had they to be redeemed by charges on the particular areas of ore which they opened up, those charges would be unfairly high.

The subject of development is further treated in Chapter XX.

CHAPTER XIV

SAMPLING AND ORE VALUATION

THE term "sampling," as applied in mining on the Rand, includes all operations which result in obtaining from any bulk of ore a smaller quantity which accurately represents the bulk in all respects except amount, and which in amount is convenient for testing, so as to enable the value or composition of the whole bulk to be ascertained.

In mining there are principally two classes of ore to be dealt with :—

(1) Unbroken ore or reef *in situ.*

(2) Broken ore.

Unbroken Ore.—Sampling of unbroken ore is done to obtain the approximate value of the ore developed or exposed. In most of the mines the sampling is done with some system, the samples being taken by the assayer, or more rarely the surveyor. In the larger mines, where it is done more systematically, special men called "samplers" are employed for the purpose, whilst in some of the smaller mines the samples are taken irregularly by various officials. In the cases of those mines which are controlled by one or another of the large financial houses, the sampling is to a great extent supervised by the engineering department attached to those houses, the result being that for each group there is a greater uniformity of the methods employed, so that the results obtained are on a better basis for comparison.

System of Sampling.—Sampling as far as possible proceeds

simultaneously with development, in relation to which the reefs may occur as under :—

(1) Wholly exposed by drives and winzes.
(2) Partially exposed by drives and winzes.
(3) Only exposed by winzes.
(4) Undeveloped.

To these divisions further reference will be made.

In most of the mines, and especially in the deep levels, samples are taken at regular intervals along all levels, winzes, shafts, etc., where ore is being exposed in the process of development.

The set distance between samples varies from 5 feet to 20 feet. The former and smaller distance is generally used in the deep-level mines, where a great deal of development work is done before milling commences. In the outcrop mines 10 feet is a more common distance.

In some mines, notably the East Rand group, the samples are taken at weekly intervals. The result is that, in consequence of unequal distances driven week by week, the samples are taken at irregular distances apart, varying with the rate of driving from a few feet up to 20 or 25 feet.

In addition, samples are taken from the stopes in the producing mines, and some from the stope drives, which in a good many mines proceed with the main drives.

The stope samples are taken to ensure that the stope face is carrying all the payable ore, and special attention is paid in taking them to the leaders in the roof or floor.

The stope-drive samples are generally incomplete, because these drives as a rule are too small to carry all the reef. Where, however, the main drive is off the reef they are very useful.

Practical Sampling.— In sampling a drive, the reef exposed on the footwall side is generally sampled, because the section on that side is the more complete, and lying higher, is not so

often covered up by water and slush. This is shown in the following sketch (Fig. 141) :—

When any reef or any section of a reef is incompletely exposed on this side of the drive, it should, if possible, be

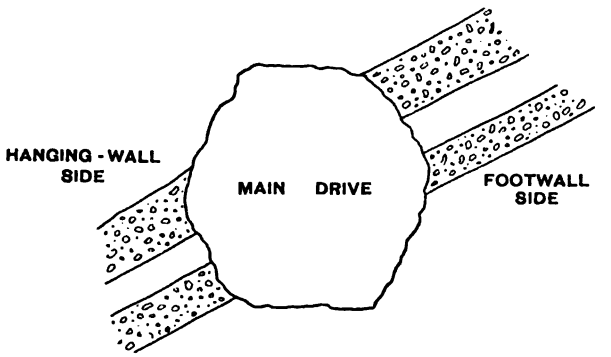


FIG. 141.—Sketch showing the usual position of the reef in a drive.

sampled from the other side. Owing to the dip of the reef the hanging-wall sections are often better exposed on the hanging-wall side of the drive, and those sections are often sampled there.

It sometimes happens that no reef is exposed by the drive. In these cases, where there are stope drives or stopes above the level, samples are taken there. These are useful, because

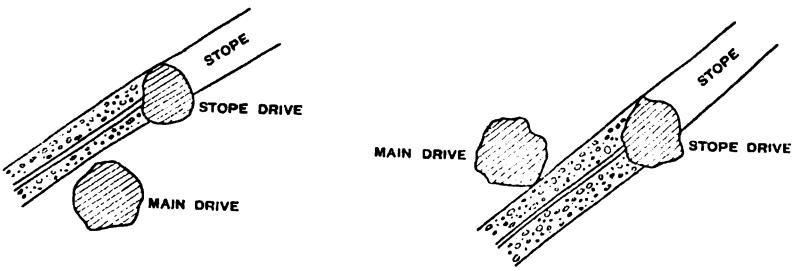


FIG. 142.—Sketches showing when samples from stopes and stope drives may be taken for computing the value of ore reserves.

though the ground may have been worked out above the level, there may be ore standing below, as is shown in the two sketches in Fig. 142.

When samples are taken at regular distances, the places

from which they are to be taken are measured off from the last survey peg. This sampling is kept up as close to the face as possible in order that the results may be up to date, and also because when a drive is old the reefs are not so well seen, everything becoming covered with a coating of mud, from the deposition of fine dust with moisture.

After the position for sampling has been measured off, the face on the footwall side is dressed down by the aid of a small pick, and further cleared, if necessary, with water, so that the different sections of the banket may be well exposed. A sectional drawing is then made giving the true thicknesses of banket and quartzite, and the condition of bedding is noted. By this means, and aided by petrological characteristics, the various sections of banket are recognised. It may be stated here that in any mine the more important banket beds are so definite and distinct that they can always be recognised by an efficient sampler. These beds form datum lines to which all the less important stringers can be referred. In some mines there are two or more leaders worked, which are distinct and separate throughout; these are named, according to their relative positions, the hanging-wall, middle and footwall leaders, and are sampled separately. After the beds have been recognised, the actual sampling is commenced. This is done best by means of a 5-lb. hammer with a short handle, and a gad. The upper banket bed is generally taken first. The gad is made to travel down the line of section, and the pieces of ore, which are broken off as small as possible, fall into a receiver, for which purpose a stiff broad-brimmed felt hat serves very well, as it is not cumbersome to carry, stands a lot of knocking about, and allows the sample to be easily transferred to the sample bag. Where the reef carries large pebbles, the gad must be made to travel down two or more lines, so placed that the largest pebble in the section can be included between them with the proper proportion of cement.

The sample bags are numbered with large black figures on the outside, and the number of the bag into which the sample has been placed is entered in the sectional drawing, against the portion of the reef sampled. The bags are made of stiff canvas. A good-sized bag is 14 inches deep and 10 inches wide when empty; on one side, about 3 inches from the top, a piece of stout tape is sewn at its middle into the seam, and by its free ends the mouth of the bag is securely tied.

In sampling a winze, that side is chosen which presents the best exposure of the reef, or the samples from the different sections are made up with pieces from either side.

It is a usual practice, when sampling a thin reef or leader which is very rich, to make the width always up to a uniform figure by including an equal amount of the quartzite on either side. This is done most frequently in the mines in the Roodepoort district, which depend chiefly on a small and rich South Reef.

In the Durban-Roodepoort Deep, when the South Reef is very thin, a sample is taken over a width of 2 inches to include the reef and quartzite. In the Banket the samples were taken uniformly over a width of 6 inches. There is a much greater chance of error in sampling a small and rich reef than a large and average grade ore.

It is not the usual practice to sample the interstratified quartzite, but where such has been proved to carry gold, as for instance that between the leaders of the Chimes Reef, that between the various sections of the Main Reef in the Central Rand, and that accompanying the South Reef on Roodepoort, it, also, is sampled.

The bulk of an ordinary sample varies with the size of the reef. It should not be less than 1 lb. and need not be greater than 6 lbs., with an average of about $3\frac{1}{2}$ lbs.

When samples are taken weekly, the faces of the drives, winzes, etc., are sampled and the distances of such points are

measured from the nearest survey peg. The procedure is, with certain limitations, identical to that previously described. Sampling from the face for purposes of the determination of ore value is apt to give results on which great reliance cannot be placed, because the face is not always squared up, and it is often dirty at the bottom. Such sampling is also apt to be done more hurriedly than it should be, because the miner hates to be hindered in his work; it is, however, most necessary during development, for the grade of the ore in the face often determines whether or not the drive is being kept on the proper reef.

Sample Grinding.—The sampler generally has a room built off from the workshops, in which the mine samples are ground up. This is best done by some form of mechanical crusher, because on the large mines there are many samples to get through. The sampling outfit which gives the best results is a Gates crusher, No. 00, fixed up to break the rock down to three-eighths of an inch, after which the further reduction necessary to pass the ore through a 60 or finer sieve is accomplished by the sample grinder shown in Fig. 143. One of these Gates crushers fixed up in the Goldfields Assay Office has been in use ten months, and the hard steel casings are hardly worn. Eighty samples were put through it once in three hours, though if it were running continuously it would do much more.

The sample grinder is often used without the crusher, in which case it is necessary to pass the ore through it more than once, and the large pieces of ore have to be broken up before they are put in. Sometimes an air drill is set up to do the crushing, the chuck end of the piston bar being beaten back to form a pestle head; this last method has not, however, been found very satisfactory, for the ore is flung out of the mortar a good deal, either as dust or small pieces, even though the top is covered over with a leathern washer. It is not so speedy as the grinders, and is more costly, for it takes

about 10 I.H.P. to run the air drill, whereas the grinder and crusher combined do not require more than 5 I.H.P.

In smaller mines the ore is broken down in large massive iron mortars by long heavy-headed iron pestles.

After it has passed through a 60 sieve, the ore is thoroughly mixed and quartered down; the resultant sample is sent in for assay and the remainder is panned off.

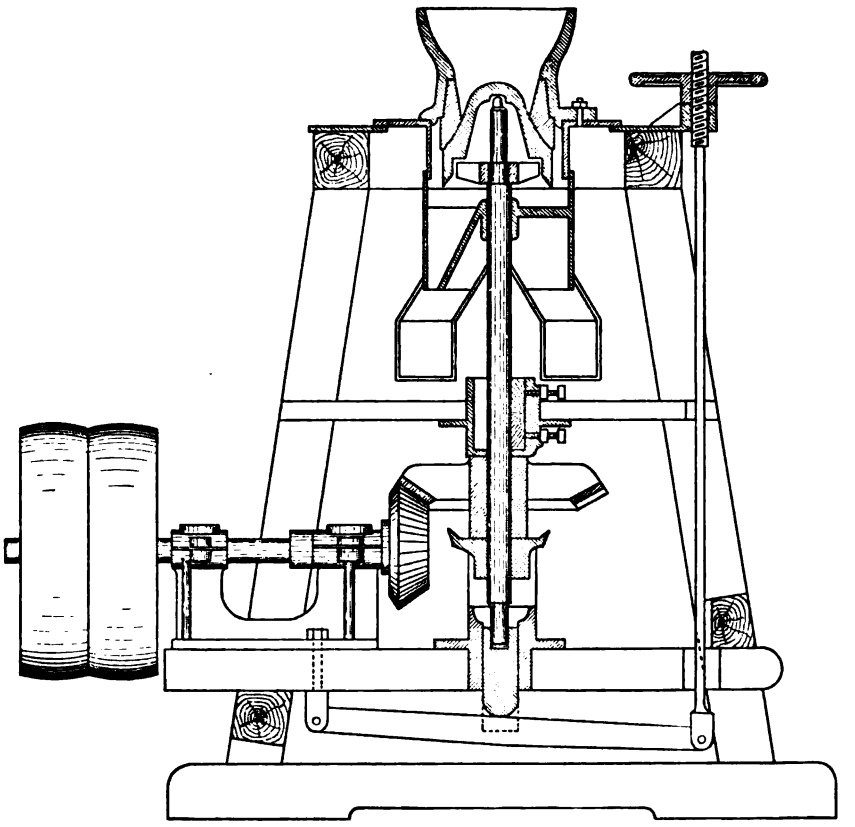


FIG. 143.—Sample grinder (Fraser and Chalmers).

Panning.—The object of panning is to get a rough idea of the grade of the ore without waiting for the assay result. This is not required in the case of those samples which form part of the scheme of systematic sampling, but where banket has been struck in a drive the officials are anxious to know as soon as possible what the grade is likely to be; and again

when, in stoping, banket occurs in the roof or floor, about which there is some uncertainty whether it be payable or not, this rough determination of the grade is speedy and reliable.

As the bulk is panned before the sample is assayed, it is well to record the panning results in order to check the assay results.

As the samples are panned before they have time to get mixed with others, where there is great divergence between the two results, it is likely that the samples sent in for assay have got mixed. In panning, the amount of pyrites is noted as well as the free gold; from these notes, and with the experience gathered from long comparison of the two results, it is possible from the panning result to approximate very closely to the assay result.

The rough-and-ready tests by the pan are very useful, but at best the results are only to be quoted in such relative terms as poor, low grade, fair grade, high grade, and rich.

It is generally found with ore from any one reef, that the more pyritic the ore, the higher is the assay result above that which would be determined from the free gold.

The old body of an iron truck, provided it is made water-tight, makes a good receptacle for the water used in panning.

Mine Sampling Records.—As soon as possible after the samples have been taken, the various particulars are placed in the Sample Book to await the assay result. The most important particulars at any point are—

1. The position along the level.

2. The complete section of the reef with thicknesses of banket and quartzite.

3. The assay result of each banket bed.

These particulars are stated in the Sample Book, in which the samples are best arranged under the respective levels in which they were taken and in consecutive order, east and west, from the central points.

On pp. 314 and 315 specimen leaves from Sample Books

are shown. In some mines, notably in the Geldenhuis Estate and the Ferreira, the different portions of the reef are marked alphabetically, the same letter always standing for the same reef or portion of reef; by this means it is possible to very closely define the particular portion of the reef which has been sampled; the only results which are fully entered up are those from the samples which have been taken over the full widths of the various definite banket beds; when any one is incomplete in its width, a portion of it lying in the roof or floor, the assay value is entered up, to give an idea of its grade, but the width is left out.

In the Geldenhuis Estate a standard section of the reef was constructed and the various portions of reef and interstratified quartzite were lettered as shown on p. 315. As the result of sampling in this mine, it was found that these various sections of the reef were especially well maintained throughout, and all the banket exposed could be recognised as belonging to one or other of them. This standard section, of course, only included the banket on which development was being done.

In the Ferreira the lettering is a little different, the bands of banket being represented by A, B, C, D, E, etc., and the layers of quartzite by A′, B′, C′, D′, E′, etc. The sampling records are there kept in books, each reef having a book by itself, marked off as shown on p. 315. The records from the Main Reef Leader and the upper portion of the Main Reef are kept in one book, as they are worked as one reef.

In addition to these sample books, most mines have assay plans on which the samples taken along the drives and winzes, etc., are placed in proper position. Where the results are distinctly arranged in the sample books according to the levels and to the drives, as in the case of the Ferreira, an assay plan is not a great necessity, but generally the final arrangement of the results is left out of the sample books to be placed on the

BANTJES

Date.	Sample No.	Assay No.	Location on Plan.	Sampled by	Reef.		Fine Gold per 2000 lbs.		Calculated Value per 2000 lbs.			Remarks.
					Name.	Width.	Ozs.	Dwts.	Stoping Width.	Ozs.	Dwts.	

ROBINSON DEEP

Reef

Level *Drive*

Date.	Reference No.	Width.	Values.			Remarks. Position and Notes.	Section.
			Ozs.	Dwts.	Grs.		

EAST RAND MINES

Reef *Level* *Drive*

Month.	Date.	Distance from Shaft.	Distance from last Sampling.	Thickness.	Thickness × Distance.	Assay Value in Dwts.	Thickness × Distance × Assay.	Remarks.	No. of Sample.

Geldenhuis Estate Sample Book

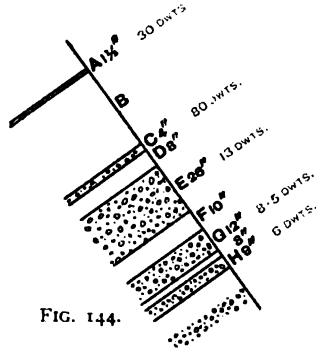
No. of Sample.	Date.	Position in Mine.	Reef.	Thickness.	Value in Dwts. Estimated by Pan.	Value in Dwts. Assay.	Section.	Remarks.
		4th Level *West Drive*, 12 feet west of Survey Peg No. 42.	A	1½″	25	30		3″ × 0 dwts. = 0
		(The average value of the thickness of banket and waste which would be stoped at this point is made up under Remarks.)	C	4″	100	80		C 4″ × 80 ,, = 320
								D 8″ × 0 ,, = 0
			E	26″	14	13		E 26″ × 13 ,, = 338
								F 10″ × 0 ,, = 0
			G	12″	6	8.5		G 12″ × 8.5 ,, = 102
								3″ × 0 ,, = 0
			H	9″	5	6		H 9″ × 6 ,, = 54
								3″ × 0 ,, = 0
								78″ × 10.4 ,, = 814

FIG. 144.

Ferreira Sample Book

Reef

Level *Drive*

Distance along Level.	A				A′			B				B′			C				D		
	Inches.	ozs.	dwts.	grs.	Inches.	dwts.	grs.	Inches.	ozs.	dwts.	grs.	Inches.	dwts.	grs.	Inches.	ozs.	dwts.	grs.	Inches.	dwts.	grs.

assay plan. A well-kept assay plan is preferable to a well-kept sample book, because when the value of any particular piece of ground is required, both the assay value and the area are to be found on it. With very few exceptions, one being the Ferreira, all the large mines keep assay plans, which are tracings from one or another of the working plans on which the thicknesses and assay values of the reef are placed in position. Where the dip is steep it is more convenient to use the longitudinal section, because the levels on the plan would be too close together to allow the figures to be distinct, but the horizontal projection is more generally used because the sampling follows the sinuosities of a drive, and these are not represented in the longitudinal section.

The best projection for the assay plan is, however, that made on to the plane of the reef, because on this all the lengths and areas are true to scale and the regularity of the sampling can be appreciated. In the case of a horizontal projection the winzes are foreshortened, and the samples taken along them appear to be too closely together, and in the case of a vertical projection (longitudinal section) neither the drives nor the winzes are represented to scale by their true lengths.

The assay plan is also generally the stope plan, an exception to this rule being that of the City and Suburban, where the assays are placed on a skeleton plan of the survey lines. When the assays and stopes are marked on the same plan it is possible to estimate not only the value of the ore developed, but also the value of that which is being mined.

It is the general practice to mark the assay value in dwts. of fine gold per ton on one side of the level in red ink, and the thickness in inches on the other side of the level in black ink; the points at which samples have been taken are indicated by small crosses, and the survey stations by small circles.

In mines where two reefs are worked separately, two assay plans are kept, and where two reefs are being worked together

their values are placed on one plan and kept distinct by proper lettering. In the Geldenhuis Estate the two reefs, Main Reef Leader and Main Reef (known in this mine as the Slate Leader and North Reef respectively), are generally worked together, and in the standard section of the reefs worked, the Main Reef Leader is lettered "C," as on page 315. The following diagram, Fig. 145, represents the method of recording the reef values on the assay plan in this mine.

A rather good method of demonstrating the presence of rich or poor ore was used on the old assay plan of the Village Main Reef. Where the reef assayed under 20 dwts. a small circle filled in with a blue wash was placed, and where it assayed over 60 dwts. a similar small red circle.

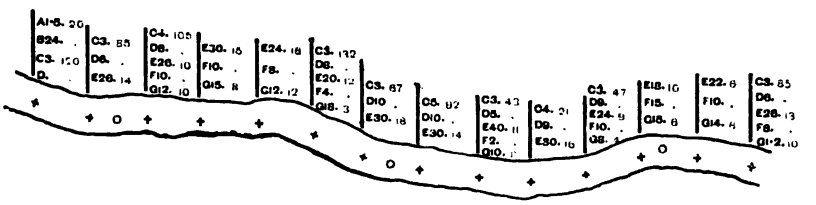


FIG. 145.—Plan of portion of a level showing the method of placing the values on the assay plan of the Geldenhuis Estate. The black figures are thicknesses in inches, the red figures are assay values in dwts. The key to the lettering is given on p. 315.

Delineation of Dykes and Faults.—This work is often entrusted to the sampler, who soon becomes a trained underground observer. The position of a fault is determined by measuring the distance from the point where it crosses the centre line of the drive, to the nearest survey peg. Its direction of strike is determined better by off-sets from the survey line than by the use of the magnetic needle. Its dip on either side of the drive is taken by means of a clinometer and the average of the two angles is taken. In the case of a dyke, the position, direction, and dip of each of its walls are determined as though each wall were a fault. The two walls having been fixed, the thickness is easily determined. On the plan, dykes and faults are best marked by indicating the line of strike on each level, with a single-headed arrow to

denote the direction of dip. The line of intersection with the reef is best dotted, as shown in Fig. 146.

When dislocation occurs, one part of the reef has been relatively down-thrown and the other up-thrown. For the sake of uniformity the position of the down-throw generally is noted, and its direction is indicated by attaching a double-headed arrow to the line of dip, as shown in Fig. 146. By thus indicating the dip and the down-throw the relation

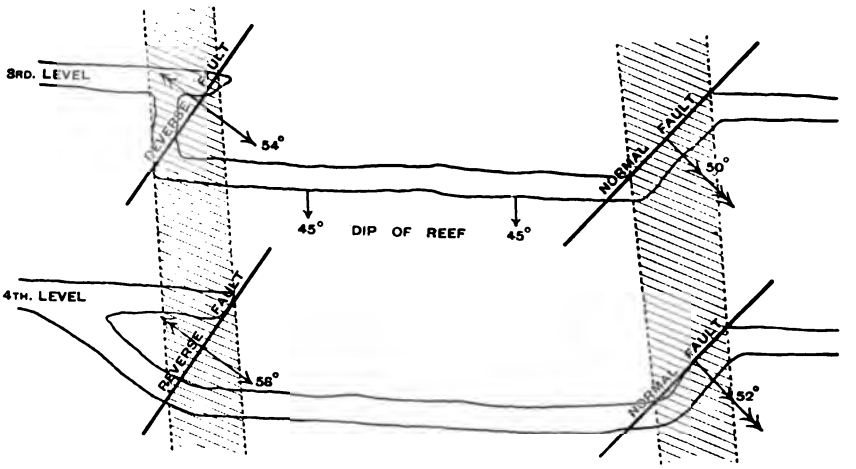


FIG. 146.—Diagrammatic plan showing the method of indicating dykes and faults in plans.

between the two is shown. Where they are on the same side of the line of strike a normal fault is almost invariably indicated, and where they are on opposite sides, a reverse fault.

Averaging Values.—The relative value of a reef or of a section of a reef at any point is represented by the multiplication of its thickness and its assay value. Thus the relative value of a piece of banket 10 inches thick and 10 dwts. assay, is 10 inches × 10 dwts. = 100 "inches × dwts."

With a composite sample at any point, made up of more than one layer of banket with some interstratified waste, it is better to enter up at each point the thickness and value of each layer, so that each is regarded as a distinct reef. If, instead, the average value of the banket exposed be entered up, there is nothing to indicate what is unexposed,

nor is the thickness of interstratified waste given. When the different layers are kept separate the value of any length is determined by averaging up the samples from each layer separately, and by then averaging the different layers in the manner shown on page 315. In that calculation the layer A and the lowest layer have been regarded as unpayable, but three inches of waste have been allowed as being stoped both above C and below H. Without the waste the value of the clean banket would be 51 inches and 15.9 dwts. It is seen that in the "stoping thickness" of 78 inches there are 27 inches of waste, of which 40 per cent, or 10.8 inches, will be unsortable fines, and 13 inches, or 16.6 per cent of the total stoped, will be about the thickness sorted out. This leaves a "milling thickness" of 65 inches, with an assay value of 12.5 dwts. The results which follow from the endeavour to keep different layers distinct are better than those from the other method at its best.

Where samples are taken at short intervals, the average between any two consecutive samples from one band of banket is taken to represent the average value between the two points at which they were taken.

Thus, if A, B and C be three points along a level at which the following reef values were obtained:

A	. .	10 inches	and	10 dwts.	=	100 inches	×	dwts.
B	. .	12	"	20 "	=	240 "	×	"
C	. .	8	"	5 "	=	40 "	×	"

then the average value between A and B, *i.e.* between two consecutive samples, is $\frac{340}{2} = 170$ inches × dwts., the average thickness is $\frac{22}{2} = 11$ inches, and the average assay value $\frac{340}{22} = 15.45$ dwts.; therefore the value between A and B is

A B . . . . . 11 inches and 15.45 dwts.

and similarly

B C . . . . . 10 inches and 14.00 dwts.

The method of obtaining an average value for lengths along which more than two samples have been taken differs according as to whether or not the samples were taken at regular distances apart.

Samples taken regularly.—If A, B and C are spaced at regular distances apart, say 10 feet, the average value between A and C is the average of the two averages previously obtained. Thus:

A B . .	11 inches	and	15.45 dwts.	= 170	inches ×	dwts.
B C . .	10 „		14.00 „	= 140	„	„
	21 „			310	„	„

and

A C . .	10.5 inches		14.76 dwts.	= 155 inches × dwts.

In obtaining this last average it is seen, by referring back, that B has twice entered the calculation, A and C only once; this is because B is regarded as holding good for 5 feet towards A and 5 feet towards C, whereas A and C, within the same distance, only hold good for lengths of 5 feet each.

If A C be extended 5 feet at either end to D and E, then in obtaining the average value along D E, the three values A, B and C enter the calculation on the same basis, for in that distance each holds good for equal lengths of 5 feet on either side and the value for D E is

Thickness . .	10 inches
Assay value . .	12.66 dwts.
D E . . .	10 inches and 12.66 dwts.

so that if samples are taken at regular distances the average value over a length, which includes, in addition to the actual distance between the first and last sample, short distances at either end each equal to half the set distance between samples, is obtained at once by averaging up the assays as just shown. For distances along which there are many samples so situated it may be taken that, neglecting the exact limit to which the end samples are regarded to extend, the simple average of all

the samples gives the value along the distance in which they are included.

The proper tabulation of the various samples gives columns which are easily kept up to date and from which the average along any length may be obtained at any time, thus :—

SPECIMEN SHEET FOR SAMPLE BOOK, WITH A CALCULATION ILLUSTRATING THE METHOD OF AVERAGING REGULAR SAMPLING

Sample No.	Distance along Level.	Inches in Thickness.	Assay Value in Dwts.	Inches × Dwts.
	Feet.			
1	1000	12	12	144
2	1010	10	15	150
3	1020	12	10	120
4	1030	13	18	234
5	1040	14	15	210
6	1050	18	22	396
7	1060	20	15	300
8	1070	16	30	480
	70	115		2034

Average thickness, 1–8. . . $\frac{115}{8} = 14.3$ inches.
Average assay value „ . . $\frac{2034}{115} = 17.6$ dwts.

These considerations apply only to complete samples. At some places it happens that, at the point where the sample should be taken, the complete reef is not exposed, or there may be no exposure of the reef at all. In such cases the value of the reef for that point must be obtained by sampling at points which are nearest in any direction and by averaging. In this way the majority of samples can be made complete.

There is another adjustment to be made when an abnormally high assay occurs amongst low ones. In some mines it would be left out altogether, but a better way is to substitute for the high assay the figure obtained by averaging it with others on either side, the number taken into the average depending upon the degree of abnormality of the assay.

Samples taken irregularly.—As with those taken regularly,

the average value between any two consecutive samples on a reef is the arithmetical mean of the two samples. Where, however, it is required to average lengths which include more than two samples, adjustments have to be made to correct the irregularity.

In most mines where the samples are taken at irregular intervals, no notice is taken of the irregularity, and the samples are averaged up as though they were at equal distances apart. This assumes that the value shown by sampling to exist at any point holds good for unequal distances on either side of that point. Such an average cannot be relied on to give a good result, for it might happen that at some part where the reef was considerably above the average value the samples occurred close together, in which case such a rich part would be unduly represented in the average, whilst the same might equally well happen in the case of a poor part.

In other mines some correction is made by the method in use at the East Rand mines, where an average is worked out as under :—

	Distance from last Sampling.	Thickness in Inches.	Inches × Distance.	Assay Value in Dwts.	Inches × dwts. × Distance.
	16 feet	12	192	12	2304
	18 „	10	180	15	2700
	15 „	12	180	10	1800
	14 „	13	182	18	3276
Totals and averages	63 „	11.65	734	13.73	10080

The average assay value is obtained by dividing 10080 by 734 = 13.73 dwts.; the average thickness by dividing 734 by 63 = 11.65 inches; and the average "inches × dwts." by dividing 10080 by 63 = 160. By this means the undue representation of rich or poor parts over large lengths is avoided, but the assumption upon which this correction depends, is that the value at any one point holds good back to the last point of sampling, or, in other words, that the value of the reef between two samples is revealed by the one

last taken. This fact makes it likely that for short distances this correction cannot be applied with success.

In dealing with areas at the East Rand mines a different assumption is, however, made. It is not considered that the average of any winze or drive gives the value back to the last one, but such value is obtained by averaging the values along the distance between the two winzes or two drives.

The most correct method of averaging irregularly taken samples is to average up each pair of samples separately, and then to average up the results, introducing each into the calculation in proportion to the length between the two samples from which it was obtained. Where more than two samples occur regularly, the arithmetical mean of this number is taken, and this result used, as are the others. This method guarantees with irregular sampling the same standard of accuracy as does the arithmetical averaging of regularly taken samples. It is illustrated figuratively by the tabular calculation on p. 324.

This method, when proceeded with on the lines indicated in that specimen sheet, is continuous; that is, the average of any length is obtained by adding up the various columns and dividing one into the other as shown; so that if a sample book be kept up to date on these lines, any average required can be obtained at once.

It is illustrated graphically in the following diagram :—

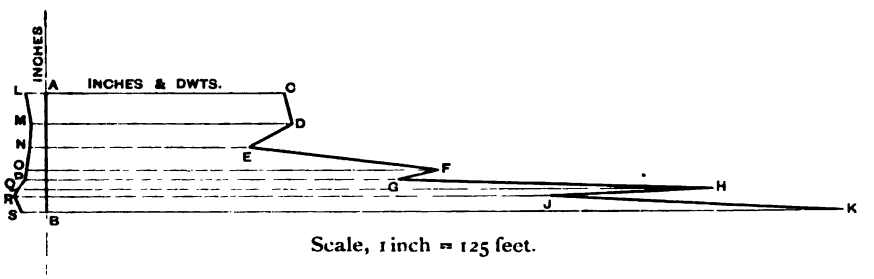


FIG. 147.—Diagram illustrating the graphic method of averaging irregularly taken samples. The same series of assays are used as those on the next page.

SPECIMEN SHEET FOR SAMPLE BOOK, WITH A CALCULATION ILLUSTRATING THE METHOD OF AVERAGING UP IRREGULAR SAMPLING

Sample No.	Distance along Level.	Thickness in Inches.	Assay Value in Dwts.	Inches × Dwts.	Number of Samples included in Average.	Average Distance between first and last Sample.	Average.			Distance × Inches.	Distance × inches × dwts.
							Inches.	Dwts.	Inches × Dwts.		
L	1000	12	12	144	...	...	...	...	...	...	...
M	1018	10	15	150	2	18	11	13.3	147	198	2646
N	1033	12	10	120	2	15	11	12.2	135	165	2025
O	1047	13	18	234	2	14	12.5	14.1	177	175	2478
P	1053	14	15	210	2	6	13.5	16.4	222	81	1332
Q	1060	18	22	396	2	7	16.0	18.9	303	112	2121
R	1065	20	15	300	2	5	19.0	18.3	348	95	1740
S	1073	16	30	480	2	8	18.0	21.6	390	144	3120
						73	13.3	15.7	...	970	15462

The average thickness of the reef from 1000 to 1073 feet along the level is determined by dividing the total distance into the total "distance × inches" $= \frac{970}{73} = 13.3$ inches, and the average assay value by dividing the total "distance × inches" into the total "distance × inches × dwts." $= \frac{15462}{970} = 15.7$ dwts.

Along the line A B the distances between the samples are placed. On one side of this line ordinates extend, that from any one point being equal in length to the "inches × dwts." obtaining at that point. The average distance of the irregular line C D E F G H J K from the line A B is the average value of "inches × dwts." It is obtained by dividing the area A C-K B by the length A B.

On the opposite side of the line the ordinates of "inches of thickness" extend, and the irregular line L-S is formed. The average thickness is obtained by dividing the area A L-S B by the length A B, and the average value by the division of this last into the former result.

It is thus seen that this method could be recorded graphically, and any result could be obtained by taking off the necessary areas by means of a planimeter. It is found, however, in practice that, owing to the irregularity of assays, it is difficult to find a scale which would suit all the values, that the lines from the winzes overlap those from the drives, and that a series of special plans has to be used. It is considered that this graphic method is good for showing at a glance where the rich or poor portions of the ore lie, but this is much better done by picking out such parts with small circles of different colours, as described before.

Determination of the Value over Areas.—So far, averages along lengths have been dealt with; those over areas are obtained by similar methods.

It is assumed in this connection that the average value around an area, when determined from a sufficient number of samples, gives a very close approximation to the average value of the area. The equable distribution of the gold throughout the banket makes this assumption reasonable; should a shoot of richer or of poorer grade ore cross any particular area, its extent is definitely indicated by the samples, and its value and extent can be separately estimated.

It may be taken that the average value of any area of

reef is obtained by averaging up those samples which, being taken only along those winzes and drives that bound the area, are also opposite to that area. This empirical rule is illustrated in Fig. 148.

Another method is to average up those samples which occur where the area borders on the drives and winzes. For instance, the value of E F G H could be taken to be the average of 15-17 and 40-43, but such a rule cannot apply in

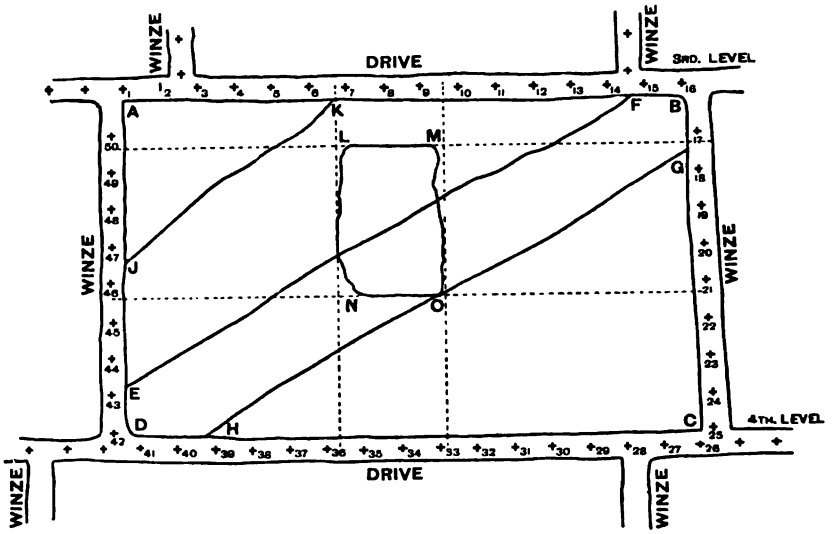


FIG. 148.—Diagram illustrating a method of obtaining a value over an area on the reef plane.

The average value of area A B C D is the average of samples 1-50.
,, ,, ,, E F G H ,, ,, ,, 1-50.
,, ,, ,, A J K ,, ,, ,, 1-6 and 47-50.
,, ,, ,, L M N O ,, ,, ,, 7-9, 18-21, 33-35, and 46-49.

the case of an area which, like L M N O, does not border on either drive or winze.

The extent to which sampling can be carried with the view of determining the value of areas on the reef plane depends upon the way the reefs are opened up. As stated before, reefs are found to occupy principally the following relations to the development:

(1) Wholly exposed by drives and winzes.

With this relationship, it is possible by complete and

regular sampling to very closely approximate to the true value of the ore developed, and in due sequence to the value of the ore being mined.

Where a reef is small and contains a lot of free and visible gold, such as the Middle Reef of the New Rietfontein property, it is not possible to get a satisfactory result.

(2) Partially exposed by drives and winzes.

With reefs which are above the average size, it is not possible in working to keep on the reef so closely that the whole of it is continuously exposed, but at different points the different portions are exposed. In these cases the samples must be made complete by placing average values, obtained from the nearest exposures, upon those sections which are not fully exposed; with such adjustments a satisfactory result should be obtained.

In some mines, notably the Simmer and Jack, the reefs are in places so close together, and the formation so flat and undulating, that the drives are sometimes on one reef and sometimes on another; in such cases the reef before sampling has to be recognised. This occurrence is similar to that of the large reefs mentioned above, but the exposures of the two reefs are not sufficient, even with making up, to give an ore value on which great reliance can be placed; when stoping commences, stope samples are necessary to ascertain whether the ore is maintaining its grade.

(3) Exposed only in winzes.

In some mines, amongst others the New Kleinfontein, Primrose, eastern portion of the Geldenhuis Deep, Simmer, New Heriot, etc., two reefs are in places close enough to be worked from one drive, but not close enough for one stope. In these cases the drive is generally on the footwall reef, which thus becomes wholly exposed, whilst the hanging wall reef is exposed by winzes only, as in Fig. 156, and these are often put up only just before actual stoping commences.

The value of the ore as obtained from sampling in these

winzes gives a good idea of its grade, but the insufficiency of the opportunities for complete sampling precludes any guarantee being given that a close approximation to the true value has been obtained.

Under these conditions stope assays are necessary to ascertain whether the ore is maintaining its grade.

(4) Reef undeveloped.

In most mines there are poor beds of banket on which no development has been done. These beds are sampled by cross-cuts, and they generally have such a small range of assay value that the average value from several cross-cuts represents a close approximation to the true values.

Comparison of Results.—In the early part of 1895, Dr. Hatch estimated that the average value of over 200,000 tons of ore reserves in the Simmer and Jack mine was about 13.7 dwts., and during the year ending June 1895 the average assay value of the ore, as sampled at the mill, was 14 dwts.

During the same year it was estimated that the value of over 200,000 tons of ore reserves in the Geldenhuis Estate was 13.8 dwts. During the year ending 31st March 1896 the gold produced showed the value of the ore going to the mill to have been about 11.4 dwts.

In October 1896 an estimate was made from stope assays of the value of the ground which had been mined and milled up to that date by the Geldenhuis Deep, with the following results:—

		Dwts.
26,572	tons of stope rock milled averaged . .	13.8
8,000	„ development rock milled averaged .	6.5
34,572		12.11

The actual value of the rock as determined from the reduction was 8.92 dwts., and as determined from the bin samples 11.70 dwts. The value of the ore mined and milled by the Crown Reef during the year ending 31st March 1896 was, as computed from the assay plan, 20.25 dwts.; and, as

computed by the General Manager from the reduction results, 12.16 dwts. fine gold.

It was stated in the General Manager's Report on the Crown Deep, 31st December 1896, that, "by the time crushing commences the average grade of the ore in readiness will yield in the neighbourhood of 11 dwts. fine gold per ton without sorting." This is equivalent to about 44s.; the actual yield from the first month's crushing was about 41s., which is being maintained.

These figures show a variety of results, some very closely agreeing with and others widely divergent from the actual results.[1]

In the case of the Crown Reef some difficulty was caused by the fact that in the upper levels, including the third, the samples were taken over what was considered a convenient stoping width, whereas in the lower levels only clean reef was sampled. It also happened that the thickness stoped was appreciably greater than that estimated, the excess being probably the poorer parts of the reef and some waste; this last is probably the reason why the estimated result is so often above the actual result.

The sampling as it is now done is a great advance on the sampling of two years ago, and with a further improvement it will probably be found to give an accurate value of the ore which is being mined and passed through the mill, with which that obtained as the result of the several processes of reduction should be in close agreement.

In the process of underground sampling, the width of the stope which will be required to take all the payable ore, and the actual proportion of waste mined, are obtained.

Trial Crushings.—In all cases the proof of the value of a reef is obtained by the value recovered per ton of ore crushed.

[1] For further results bearing on this point see the footnote at the end of this chapter.

In some cases this method of testing the value of a reef is tried before further development on it is commenced.

In the Simmer and Jack during October 1893, twenty stamps were run on deep-level ore to try it.

In the Robinson a special test of the Main Reef was made under ten stamps, for which purpose the drives in the lower levels which were being run on the Main Reef Leader were turned into the Main Reef for a time.

In the Agnes Munro, a trial crushing of 1000 tons of ore from the second level was made with the following results :—

	Gold got.	
Mill . . .	670.30 ozs.	13.40 dwts. bullion per ton crushed.
Cyanide . .	105.00 „	2.10 „ „ „
Concentrates .	52.80 „	1.05 „ „ „
	828.10	16.55

A test crushing of the Blue Sky Leader, near the western boundary of that property, was made during 1894, when 1648 tons crushed yielded from all sources 18.5 dwts. of bullion per ton crushed.

A similar test was made of the Main Reef in the Meyer and Charlton, 56 tons of which, put through the mill, yielded 6 dwts. 21 grains of bullion on the plates per ton crushed, and the tailings assayed 5 dwts. 8 grains. This ore was mined from a drive embracing the whole width of the reef for about 12 feet.

When such a method is used the results are better returned as dwts. of fine gold per ton crushed, for bullion represents no absolute value, unless its fineness is given at the same time.

Sampling of Broken Ore.—On the surface, after the ore has been sorted and crushed, an attempt is generally made to get at the value of the ore which is about to be milled.

This is generally done before the ore is dumped into the mill bins, either at the shoots from the crusher bins, or along the track from the crusher station to the mill; no mechanical

sampler is used, a native taking off a portion of ore from each truck with a shovel. The amount so collected is broken down small, either by the native whilst he is waiting for another truck, or in bulk at the end of the day; by this means as much as 2 cwts. is often collected per day. This is well mixed and, without being ground fine, is quartered down till about 10 lbs. remain, which is crushed fine and used as the sample.

The results so far obtained by these means have been generally untrustworthy.

In the Kleinfontein such samples are taken of the ore, both as it leaves the crusher station and as it arrives at the mill, and the results obtained are most unreliable, though about 3 cwts. are taken in a day.

In the Angelo the results from the South Series are better, while with the North Series of the Comet they are generally some dwts. too high, but sometimes too low.

In the Geldenhuis Deep the following figures show the unsatisfactory nature of the results :—

Month.	Gold Contents of the Ore per Ton in Dwts. of Fine Gold as deduced from		
	Crusher Samples.	Battery Bin Samples.	Mill and Cyanide Result.
January 1896 .	12.1	9.9	8.4
February ,, .	10.0	10.1	6.9
March ,, .	12.6	9.8	8.3
April ,, .	8.1	8.3	8.0
May ,, .	9.8	10.3	7.8
June ,, .	12.9	10.6	9.3
July ,, .	...	12.8	9.4
August ,, .	...	11.5	8.2
Sept. ,, .	...	10.8	7.6
Averages .	...	10.44	8.21

The values in the last column were obtained by considering that the extraction obtained by mill and cyanide was 80 per

cent of the actual value. It would appear that, as a rule, these results are too high. If the ore were sampled before passing through the crushers there is no doubt that the results would be even less satisfactory. In the case of the George Goch, it is possible to sample the ore after it has been stamped, because no amalgamation of the gold goes on inside the boxes, and much better results are thus obtained. At this mine samples are taken once an hour from the lips of all the mortar boxes, and from these one daily sample is taken for assay, with the following result :—

During the year ending 31st August 1896, 98,017 tons of ore were milled, and the average of 533 such daily samples was 8.69 dwts. On this basis the tonnage crushed contained 42,590.70 ozs. of fine gold.

The following results were obtained by reduction :—

	Total Fine Gold.		
	Actually recovered.	Accounted for by Assay and Tonnage.	Dwts. per Ton milled.
	Ozs.	Ozs.	
Mill . .	21,854.10	...	4.46
Cyanide . .	12,322.81	...	2.51
Slimes . .	...	3838.96	0.78
Residues .	..	3619.86	0.73
Slags . .	500.00 [1]	...	0.10
	34,676.91	7458.82	8.58

[1] Estimated.

In this case the value calculated from the reduction of the ore, 8.58 dwts., is seen to be in very close agreement with the value as determined by sampling the stamped ore as described above, viz. 8.69 dwts. If, instead of building up the value in this way, the results from the mill and cyanide alone be considered as an 80 per cent extraction, then the actual value of the ore works out at 8.7 dwts., which figure is also in very close agreement.

CROWN REEF GOLD MINING COMPANY

Statement of the Tonnages of Ore milled and treated, and of the Amounts and Percentages of Gold won during the Year ending 31st March 1897

	Tonnage.		Percentage of Ore leaving Mill.		Total Gold in Fine Ozs.		Fine Dwts. Gold per Ton.		Percentage of Gold Contents.	
			Treated.	Lost.	Recovered.	Lost.	Recovered.	Lost.	Recovered.	Lost.
	Tons.	Tons.	Per cent.	Per cent.	Ozs.	Ozs.	Dwts.	Dwts.	Per cent.	Per cent.
Mill	199,898	...	...	...	67,939.693	...	6.797	...	50.010	...
Concentrates . . .	...	7,959	3.981	...	10,533.253	...	1.053	...	7.753	...
Concentrates residues .	...	...	...	...	...	1,808.888	...	0.180	...	1.331
Sands	...	137,408	68.739	...	30,815.802	...	3.083	...	22.683	...
Sands residues . .	...	...	...	...	...	10,786.268	...	1.079	...	7.939
Slimes	...	21,539	10.774	...	3,101.241	...	0.310	...	2.282	...
Slimes residues . .	...	...	...	...	...	1,446.124	...	0.144	...	1.064
Slimes not treated . .	...	32,992	...	16.504	...	9,418.555	...	0.942	...	6.933
			83.494	16.504	112,389.989	23,459.835	11.243	2.347	82.728	17.267
Totals . . .	199,898	199,898	99.998		135,849.824		13.590		99.995	

Determination of Ore Value by Reduction Results.—When a certain tonnage has been milled its actual value before it was milled is very closely estimated by building up the results obtained during reduction.

The chemistry of the retreatment process is now so well understood, and the bulk of the ore after it leaves the mill so well accounted for, that the amount of gold existing in any small untreated portion and in residues can be determined by assay. The full amount actually recovered, with the amount thus estimated as unrecovered, make a total which is a very close approximation to the amount which was originally in the ore, but which is open to the objection that it is vitiated by any loss which may arise from gold thefts.

The table given on p. 333, which is taken from the Crown Reef General Manager's Report, 31st March 1897, illustrates perfectly this method of putting a value on the ore.

It is the rule for most mines to accept such an estimation in preference to a determination by sampling before the recovery of the gold by reduction commences, the notable exceptions being the George Goch, which has been instanced before, and the Ferreira, both of which accept the original assay value.[1]

[1] The following table giving the estimated values recoverable—by the reduction processes with sorting and the slimes process—as they were obtained in the most important of the deep levels by sampling before crushing commenced, and giving also the actual value recovered during the first five months of 1898, shows the close agreement between the estimated and the true values which points to the great importance and service of sampling.

Mine.	Estimated value recoverable. Shillings per ton milled.	Actual value recovered. Shillings per ton milled.
Crown Deep	48	39.38
Robinson Deep	60	63.10
Nourse Deep	48	47.45
Jumpers Deep	35	43.42
Geldenhuis Deep	36	38.81
Rose Deep	40	44.30

CHAPTER XV

STOPES AND STOPING

Stopes.—A stope is a working underground, in which a section or block of reef which has as far as possible been opened up by levels and their connections, is being removed.

The term stoping includes all the methods and operations which result in breaking the solid reef and placing it in position ready to be trammed away.

In all the stopes which are being worked in ground which has been fully developed, it is possible by a proper arrangement of benches in the working face to put in shot-holes which are so inclined that water may be retained in them, with advantage in keeping the bottom of the hole clear. This implies that to a great extent the blow which is used in stoping is always a more or less downward one; this being the case, stopes cannot be classified by the direction of the blow; they are classified by the relation of the working face to the position from which the ore is being attacked, and are thus separated into two main divisions:—

(1) Underhand Stopes.
(2) Overhand Stopes.

In an underhand stope the ore which is being removed is below the men as they are drilling, and in an overhand stope it is above them.

The nature of a stope is, however, best defined by the relation of the direction of the working face to the winze from which the stope was started. The face of an "underhand

stope" converges towards the winze going downwards, as shown in Figs. 149, 150, and 151; and that of an "overhand stope" converges going upwards, as shown in Figs. 149 and 152. By this relation three other classes of stopes are defined, viz.:—

Combined Stopes.
Breast or Side Stopes.
Longwall Stopes.

"Combined stopes" are those which in the lower portion are overhand, and in the upper portion are underhand, as shown in Fig. 154.

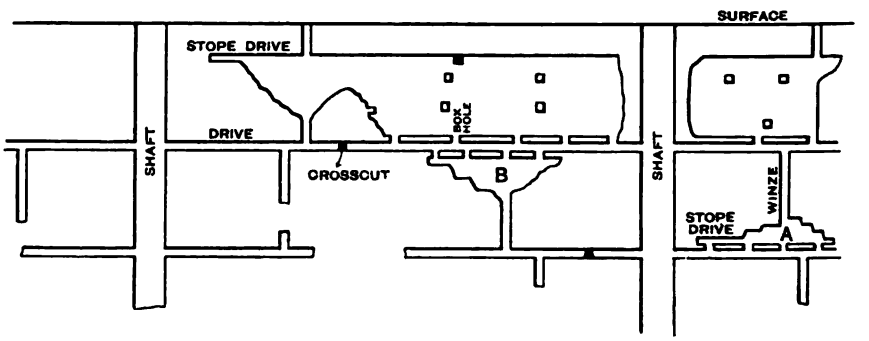


FIG. 149.—Diagrammatic longitudinal section of a mine, showing methods of stoping and the plan of development.

A, an overhand stope.
B, an underhand stope.

"Breast or side stopes" are those where the working face is more or less parallel to the winze, and "longwall stopes" are those where, the reef being approximately horizontal, the working face is advanced parallel with a line of development.

Underhand Stoping is used in its truest sense in those portions of ground which are being worked without a connection to a level beneath. It frequently happens that a piece of ground is limited in its extent below a level by dyke occurrence, as shown in Fig. 153, through which it would not be economical to make connection with any level below, to work such a piece. Should the extent warrant it, an inter-

mediate drive may be put along the intersection with the dyke; but where the piece is small, it is better to break it and to pass the ore up to the level above, with the aid of a windlass or by shovelling. Underhand stoping in its general sense is used where the reef dips steeply, as in the upper levels of the mines in the Central Rand. In such stopes it permits of a more downward blow, and it is safer because any pieces of rock which are broken from the face have not a free fall.

Owing to the steepness of the reef, the broken ore is easily shovelled along the face to the box-hole at the bottom,

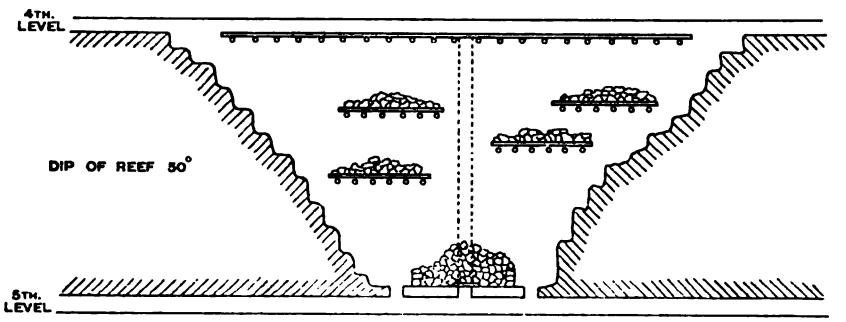


FIG. 150.—Diagram showing the Crown Reef underhand stopes.

through which it is passed down to the level below. Such a stope is shown diagrammatically in Fig. 150.

These stopes are opened by first taking off the corner of each side of the bottom of the winze, so as to get a box-hole in on either side, then from the top of the winze the working face is opened out in the manner indicated in the diagram (Fig. 151) by the dotted lines. When the face has reached such a position as A B, further work is stopped at the top, in order to get the bottom portion of the face advanced to the position indicated by the dotted line C D, when another box-hole is put up, and the top portion of the working face is again advanced. These box-holes are usually about 25 feet apart.

With this method it is usual to stope away all the rock

right to the upper level without leaving a pillar there; along the lower level, either a pillar is left through which box-holes are made, or the whole of the ground is taken away and protection is afforded to the level beneath by putting in pieces of timbering called "stull pieces," across which lagging is placed, and on which waste is packed to form a "stull." Should a pillar be left along the level at the "heel" of the stope, no tight corner is caused, because the ore can be broken in benches right down to the pillar.

It is sometimes found in working oxidised ore that it is necessary to leave pillars along the top level, in which case a tight corner would be formed at the toe of the stope. In

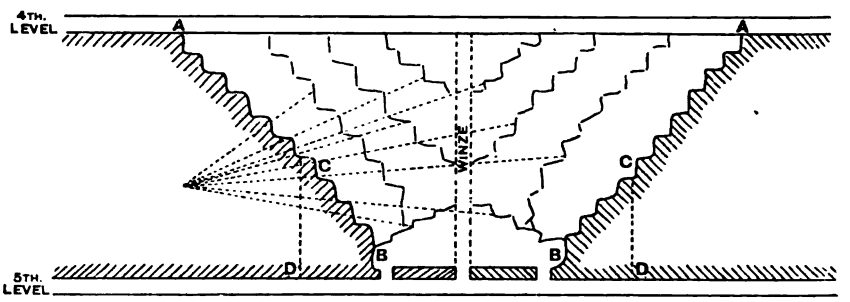


FIG. 151.—Diagram showing the method of opening out an underhand stope. The toe of the stope is at A and the heel at B.

order to relieve this tight corner it is usual to keep a stope drive going ahead at this point, as shown in Figs. 149 and 154. This is being done in the oxidised ore of the Angelo, where the reef dips about 50°, and where underhand stoping is being principally employed.

Where the waste sorted out from the face is packed in these stopes, it is necessary to place stull pieces up in the stope, on which lagging and waste rock may be placed; these also serve to protect the natives at work from anything which might fall from the level above. It is necessary where no pillars are left, as is usual in the unoxidised rock (spoken of as the "blue" rock), to place stull pieces and pack a little waste along the top of the stope. Where the dip of the reef is less

steep, this method of stoping loses all its advantages owing to the difficulty of shovelling the broken ore along the face to the ore-shoot and bore-hole.

Overhand Stoping.—Where the reef dips steeply, this method of stoping is not so largely employed on the Rand as the former method, because the native can sit comfortably to his work and strike a good downward blow in an underhand stope, whereas with an overhand stope he has to build his seat up, and occasionally has to put in a hole which has an upward tendency, at which he is not good. In the less steeply inclined stopes, say 35° and less, these objections to overhand stoping are lessened, and this method has then

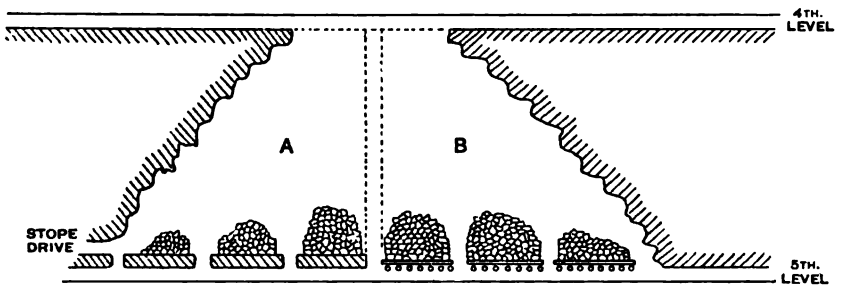


FIG. 152.—A and B are overhand stopes. In A, a stope drive and pillars along the lower level are being used; and in B, stulls along the lower level, with pillars at long intervals.

the great advantage that the broken reef drops away from the face down towards the boxes in the level below. As most of the lower levels of the outcrop mines dip from 30° to 35°, and practically all the deep levels dip about the same, this method of stoping is in greater use than underhand stoping. Another advantage pertaining to it is that the working face is generally opposite to two or three boxes which can all be in use at once, whereas with underhand stoping it is generally the case that only one box—that at the heel of the stope—is in use at any one time. This being the case, fewer box-holes are required in overhand stoping, and with an ordinary-sized reef they are generally placed about 30 feet apart.

The above diagram (Fig. 152) shows the general outline of one of these stopes.

These stopes are opened out on either side from the bottom of a winze, except when they are made in such tight pieces of ground as are limited in their upward extent by a dyke, through which it would not be economical to connect up with the level above, in which case the stope is started from the level, and no winze is used. Such a piece of ground is shown in the diagram in Fig. 153.

In an ordinary block of ground the stope face is advanced from the bottom of the winze until it breaks right into the level above, along which a pillar is not left, except in some places in the red (oxidised) ground. Along the lower level, either a pillar is left through which the box-holes are made, or the stope face breaks right through into the level. For the protection of the travelling way underneath, and to keep it clear, it is necessary in this latter case to set up stull pieces and packing and to wall up the waste so that passes for the ore to the boxes are kept open. These two methods are shown in Fig. 152, at A and B respectively.

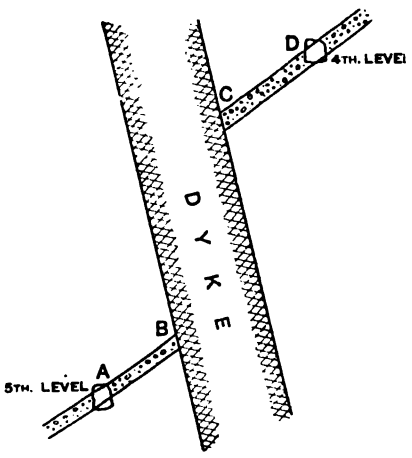


FIG. 153.—A B is a piece of ground which would be worked by overhand stoping, and C D by underhand stoping without connection by a winze.

The choice between the two methods depends greatly upon whether the value of any pillar of ore is likely to cover the cost of the timber which would be required in its place, in addition to the cost of mining and reduction. In most cases with ore of average grade, considering first the solidity which pillars give; secondly, the fact that at the last it will be possible to extract some of them; thirdly, the absolute protection afforded to the drive underneath; and fourthly, the large amount of backs which are now generally found between two

levels, it would appear to be the better plan to leave a pillar along the lower level of a stope. When this is done, the toe of the stope is a tight corner, which is either relieved by keeping a small stope drive some few feet ahead of the working face at this point, or by running a stope drive about 6 feet above the main drive during development. In most of the deep levels, where, when the producing stage is reached, the stopes are required to yield a large amount of ore at once, it is usual to make these stope drives shortly after the main drives are complete and they are charged to development. As, whenever this drive is made, it costs the same, it would appear to be the better plan to get it over at once during development, for it would then give better opportunities for complete sampling and more correct valuations of ore reserves could be made, because the stope drives are in all cases kept on the reefs, whereas the main drives are sometimes off the reef.

Where the stope face is continued so as to break through into the lower level, stull pieces covered with lagging and waste are kept along the level up as near to the toe of the stope as possible, and pillars are left at long intervals. A level along which such stoping is to be done is often prepared by taking out the back of the level with machine drills, and by making the stulls and box-holes before the stope face is advanced.

Where the reef is flatter and the ore does not fall far from the face in the stope, it is not necessary to protect the drive so much; in these cases it is a common practice to leave, alternately, equal lengths of pillar and of open space without stulling; in these open spaces the boxes are arranged.

In overhand stopes, where the large waste is sorted in the stopes, it is packed up from the lower level, the passes to the box-holes being kept open by rough walling; no stulls are required higher up in the stope, as would be the case in underhand stopes.

Combined Stoping.—In this method of stoping, the lower portion of the face is worked as an overhand stope and the upper portion as an underhand stope, so that a combined stope has two toes, one bordering on each level, whilst the heel is at the junction of the two stopes. The proportion of the two depends upon the dip of the reef; thus in the Angelo with a dip of about 55° the underhand stope is in greater proportion, and in the Princess with a dip of under 30° the overhand stope is in greater proportion; these two cases are shown in Fig. 154.

With a high dip the underhand portion has all the

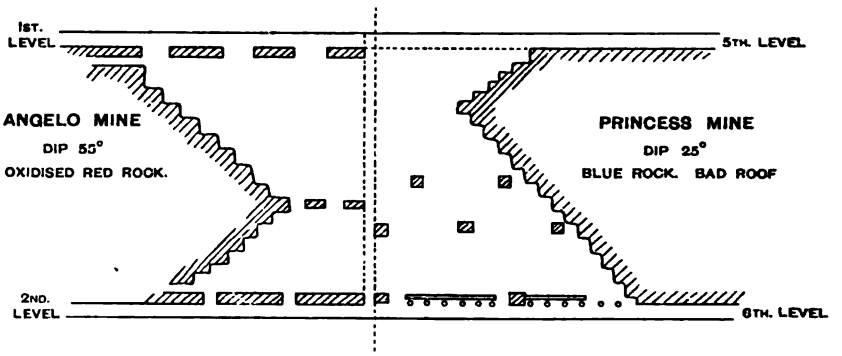


FIG. 154.—Diagram showing combined stopes.

advantages of underhand stoping, to which may be added the greater facilities for packing waste in the overhand portion, whereas with low dips the underhand portion permits a good deal of the ore to be thrown up to the upper level instead of being passed down the greater length to be trammed away from the lower level. Another advantage is that with long backs it is often advisable to leave a pillar in the centre of the stope, and this can be conveniently and inexpensively done at the heel of the stope.

Generally, and especially for long backs, this method of stoping is a good one, and it is on this account very extensively employed.

Breast or Side Stopes.—These stopes are sometimes

found where the reef is flat, but they are generally only an accidental variety of the overhand stope, as the parallelism to the winze is not generally maintained long.

Longwall Stopes.—These are used in some mines on the Black Reef, notably the New Midas Estate, where the reef is practically flat; a line of rails can be placed parallel with the working face and far enough away to be out of danger from the blasting, and along this the trucks can be brought up to take the ore away.

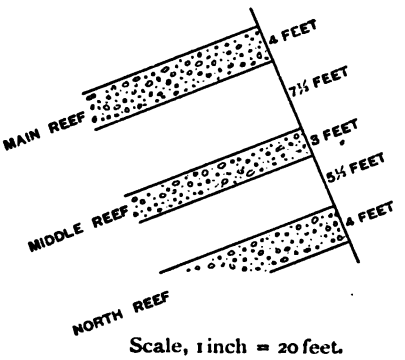


FIG. 155.—Section of the three reefs in some portions of the western section of New Primrose where, for a limited extent, there is a stope on each reef.

Methods of working Reefs which are close together.—In some of the mines two reefs are close enough together to be worked from one drive but not from one stope; in these cases a separate stope is made on each reef. In some places in the New Primrose three such stopes occur close together, the reefs lying in the positions shown in Fig. 155, where the names by which the reefs are known in the mine are given. Here the lowest stope is always kept in advance, and frequent pillars are left to support the floor of the stope above.

In the New Kleinfontein the two stopes, on the Main Reef Leader and Main Reef, are on an average separated by 10 feet of quartzite. The leader stopes, which are on top of the others, are opened out from winzes on the reef plane which have been made from cross-cuts to the reef from the Main Reef drives. The ore broken in the leader stopes is passed through its own box-holes and shoots down to the drives; the following section shows the disposition of these two stopes.

In some parts of the George Goch the Main Reef

Leader is so far away from the Main Reef that separate stopes are required; of these, the Main Reef stope, which is the lower, is kept ahead; in stoping the leader, holes are made down through the floor into the Main Reef stope, or sometimes the leader has its own boxes on the hanging wall side of the drive. The winzes are all on the Main Reef, and when one of these lower stopes is finished, each winze is broken up through into the leader, and stope faces are advanced from it on either side along the plane of the leader.

In a good many mines it happens that two reefs have to be worked in one stope because the thickness of quartzite

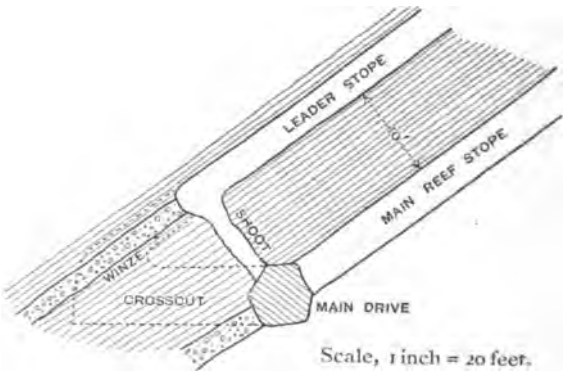


FIG. 156.—Diagram showing the method of working the two stopes in the New Kleinfontein.

between them is not sufficient to stand safely when the reefs have been taken out. In these cases, where the stope is very large, it is best to take out the reefs in benches, starting with the top one first, as in Fig. 167; this is the best way, especially with a flat dip, because after the lower reefs are out, it would be difficult to get up to the top reef without the help of considerable packing, on to which the reef would be blown and a good deal lost. In some mines, such as the Geldenhuis Estate, the top reef is a rich and small one, and the underlying reef is large and of lower grade. In these cases the lower reef is often taken out first and the richer leader dropped afterwards; by doing this, advantage is taken of the strongly developed plane underneath the upper reef.

Width of Stopes.—The average width of stopes along the Witwatersrand is about 5 feet; this width depends, first, upon the size of the reef; secondly, upon the dip; and thirdly, upon the development of the bedding planes near the reef.

In the New Primrose and Langlaagte Estate the stopes are sometimes as much as 15 feet wide, owing to the large size of the reefs. In some mines of the Central Rand, where the South Reef is split up into two or more leaders, large stopes are worked on this reef; thus in parts of the Bonanza a width of 13 feet has been reached. Where the reefs are very small, amounting only to compact leaders, it is endeavoured to obtain as small a stope as possible, but experience has shown that the smallest stope obtainable at the average angle of dip along the Rand is about 30 inches, if measured carefully.

On the South Reef in the West Rand mines the miner in charge of a stope which is measured by the sampler to be over 2 feet wide is dismissed; the measurements are, however, taken right in the face where they are likely to be less than if taken in the body of the stopes, for there is a considerable amount of rock which flakes off from the floor and roof after the stope face has been advanced.

In the Roodepoort district, where the mines are working principally the small and rich South Reef, which is only about 5 inches thick, the average size of the stopes is about 36 inches.

The Chimes Reef in the Van Ryn district consists of two or three leaders which have a total width of about 6 inches of banket, but which are interstratified over a width of about 40 inches, necessitating a comparatively large stope.

Where a reef dips steeply it is possible to work comfortably in a stope of small width, but where the reef is nearly flat larger stopes are a necessity for economical working.

In consequence of bedding planes above and below the reef, false backs and floors often occur, up to which the

reef breaks in being blasted, and these make the stope larger.

In some cases the width of a stope depends upon the nature of the rock in which the reef is enclosed; for instance, the Nigel Reef has an average thickness of about 8 inches, and in its footwall there is a soft slate in which the stope face is first advanced, the reef being blasted down afterwards. In consequence of this method of stoping, which was designed to take advantage of the soft slate in the footwall, the ultimate breadth of the stope is about 36 inches. A similar method was used in stoping the Blue Sky Leader. In the New Midas Estate the Black Reef has for its hanging wall a friable slate rock which is always mined in advance of the reef itself, so that the size of the complete stope is out of proportion to the size of the reef.

Supporting the Roof.—The roof is supported (1) by pillars, (2) by stulls, (3) by timbering.

Pillars.—In describing the methods of stoping, the use of pillars in protecting the level from any broken ore falling from the stope face was shown. Their greatest use, however, is to support the roof. The number which has to be left depends, first, upon the nature of the roof, whether it be broken or disturbed, or whether there are bedding planes along which the roof is apt to split off and fall away; secondly, upon the dip, for with a flat dip the roof is naturally more heavy than with one which approaches the vertical; and thirdly, upon whether the stope is in the oxidised or unoxidised ore.

The pillars used in stoping may be roughly divided into level pillars and stope pillars.

Level Pillars.—The position of these pillars has already been described, and they are shown in Figs. 149-152 and 154, so that a short *résumé* only is necessary here. They are long pillars, about $5\frac{1}{2}$ feet wide, which run along the levels. It is only in oxidised ore that such pillars are regularly left along the upper level of a stope, that is, along the lower side of a

drive, but along the lower level of a stope, that is, along the upper side of a drive, they are generally left, because there they efficiently serve to support the roof, and, in addition, they protect the drive from anything which might fall from above. In the New Kleinfontein oxidised ore, pillars are left along either side of the drive; those on the upper side are cut through about every 40 feet for box-holes to the stope above, and those on the lower side about every 30 feet, which is considered to be the limit to which they can be robbed with safety until the level is no longer required.

Where the inclination of the reef is such that there is little to fear from falling rock, pillars, left standing alternately with open spaces of equal length, are sufficient to support the roof in the blue ground.

Where a stope drive proceeds simultaneously with development, care must be taken that it does not draw too near to the main drive, otherwise, in blasting, the pillar between the two would be weakened; its minimum thickness should be 5 feet.

Winzes to be used for air or travelling ways are also especially protected by leaving pillars along them. In opening out stopes from winzes it is a common practice to cut round a pillar with a machine drill on either side of the top of the winze, in addition to the pillars which are almost invariably left at the bottom.

Stope Pillars.—These serve only as supports for the roof in a working stope, and their number depends upon the character of the roof. Where it is bad they are placed in zigzag fashion, as shown in Fig. 154, at an average distance of about 40 feet apart, but where the roof is good and level pillars are used, stope pillars are then only left in those places where either the reef is poor or the ground locally disturbed. As with the level pillars so with these, it becomes a question whether the gold in any pillar will pay for the cost of the timber required to replace it, in addition to the

mining and reduction expenses which would be incurred in its extraction. Where the reef is rich and the stope a small one, as in the Durban-Roodepoort, very few stope pillars are left, whilst those left along the upper level are drawn just after the stope has been swept down, and those along the lower level remain till afterwards. It is the experience in some mines that where machine drills are used in the stopes more pillars are required. It would appear to be usual to leave pillars chiefly in the lower part of the stope, because most work goes on there, and, they are more accessible when the process of robbing is commenced. Where the roof is regular the pillars are placed regularly, but wherever a break occurs the roof must there be especially supported.

Where above the reef there are false backs which fall in stoping, these may be caught up by a row of pillars along the break, and after once having been caught up they can be kept up by having a pillar here and there.

The average size for stope pillars is about 7 or 8 feet square.

The great use, then, of the pillars described is to keep the roof up whilst the ground is being worked. If left for some time they shell off and fret away owing to weathering, and become less strong, so that they occasionally give way under the weight pressing on them, but by that time their purpose is served. As soon as in any part it is no longer necessary to keep the roof up, these pillars are taken out as far as is possible, starting with those which are near the boundaries, and coming towards the shaft. This process is termed "robbing" the pillars. In addition to those already described, there are shaft, boundary, and other safety pillars.

Shaft Pillars.—Where an incline shaft has been sunk on the plane of the reef it is necessary for its preservation that pillars of ore be left, about 10 or 12 feet wide, one on either side of the shaft. These are only to be broken through as occasion demands. Similarly, but not to the same extent,

if the shaft is above the reef, some pillars have to be left immediately beneath it, whereas if it be sunk below the reef, no such protection is required.

The base of a vertical shaft has to be protected in the same way as long as the shaft is in use or kept in repair.

Boundary Pillars, etc.—It is stated in the Mining Regulations that on the inside of the boundary lines of every metalliferous mine, safety pillars must be left standing, not less than 10 metres in thickness measured at right angles from the boundary line, and that these pillars shall not be weakened, cut through, or worked unless by express agreement between the owners of such adjoining mines, or with the written permission of the State Mining Engineer, to be obtained through the Inspector of Mines.

For the protection of objects on the surface specified in the Mining Regulation, it is stated that "Mineral deposits shall be left intact not only vertically below the same, but for such distance outside the boundaries of the ground or surface objects as the Inspector of Mines may consider necessary. . . . The weakening of these safety pillars by partly working them or the entire removal of them with the object of winning mineral is only allowed with the special permission of the State Mining Engineer." Similar permission has to be obtained to drive levels through such pillars for the purpose of connection. There are very few mines which are affected by these necessary regulations, for most mines own the surface as well as the mineral rights. Where, however, railway lines cross any property near the outcrop some provision for safety is enforced.

Stulls.—Stulls are accumulations of waste rock systematically packed in areas which have been worked out. Where such packing is carried up on pieces of timber, such pieces are called "stull pieces."

Stulls are rarely used to support the roof in a working stope; they principally serve as places where waste rock

sorted out from the face may be stowed, and when the pillars give way or are robbed, the stulls take up the weight of the roof and prevent it from coming in entirely.

It is customary now, with efficient sorting plants, to send as much as possible of the rock that is broken to the surface to be sorted, but where a lot of waste is broken the larger pieces are sorted out underground. This is especially the case with small reefs, or with those reefs, such as the South Reef of the Central Rand, which often consist of several leaders separated by larger thicknesses of quartzite.

With underhand stoping it is necessary at the start to build up stulls in the stope, as shown in Fig. 150, because the base of the area worked out is not large enough to take all the waste. When, however, the area worked out is considerable, the stulling can be done up from the lower level. With overhand stopes the stulls are built up from the lower level either on the pillars or on the stull pieces; passes to the boxes through such accumulations are made and maintained by rough walling, as indicated in Fig. 152. Wherever stulling is done, it is found that some pieces of ore get mixed in with it, though this occurs to a greater extent in overhand stoping, for the broken reef in many places falls on to the stull.

In the rich mines especial care has to be taken that the stull is out of the line of fall from the working face, for it is always the finer and richer portion of the ore which is the more likely to get down on to the stull.

In the Robinson mine it was endeavoured to minimise this by placing canvas on the stull, but this got cut to pieces and the method had to be given up, whilst some stulls had to be picked over again. In some cases there were two advancing faces in the stope—first, the ordinary working face; and, second, the stull face, as shown in the following diagram (Fig. 157). Other precautions taken are to sweep the floor down before it is covered up by stulling, and

to build the waste as quickly as possible right up to the roof to minimise the area of the stull on to which the fines may fall.

The greatest part of the stulls underground have, however, been sent down from the surface, being clean waste thrown out on the sorting floor. It is generally arranged that the waste dump from that floor is run out over a winze or old shaft, so that the waste runs down by gravity to whatever level it is required in, when it is trammed and tipped into those areas which are ready for it. This is an inexpensive way of

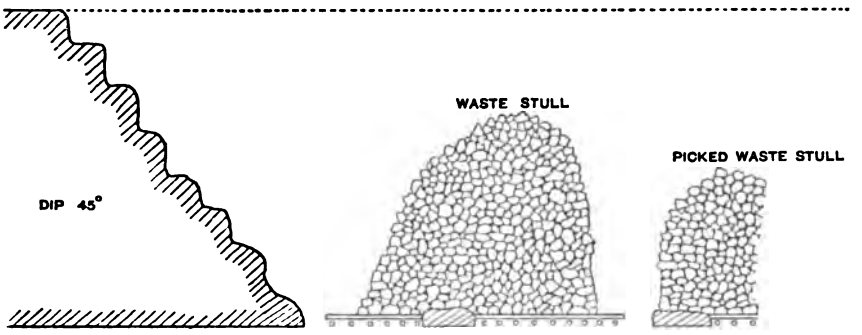


FIG. 157.—Diagram illustrating the procedure in picking over a stull.

filling up the worked-out areas, especially in those mines which have a steep angle of dip near the surface. Such mines are especially favoured in this respect, because only the upper levels require filling, for should a subsidence of the roof in the lower levels occur, it would not reach to the surface.

In the lower levels, where the dip flattens so that the waste for packing will no longer run, the waste rock cut during development, in addition to the waste picked out underground, is carefully stulled in worked-out areas.

It is necessary that the stull pieces be very firmly fixed, for a stull when opened runs like a stream. They are placed 5 feet apart; each piece is slightly tapered at its bottom end, and inserted in a hitch which has been chiselled out, an inch

or so deep, in solid rock, from which all flakes have been removed. After insertion it is further secured by driving in small wedges of wood, and when it is being put in the top end is beaten down into position from the upper side, from which also the weight will ultimately come. It is not usual to cut a hitch in the roof, but the length of the stull piece is made slightly greater than the distance between roof and floor, so that any weight from above tends to fix the piece more firmly. Upon these pieces lengths of smaller lagging are spiked close together, and large pieces of rock are built up against the barrier so formed, as illustrated in the following diagram (Fig. 158).

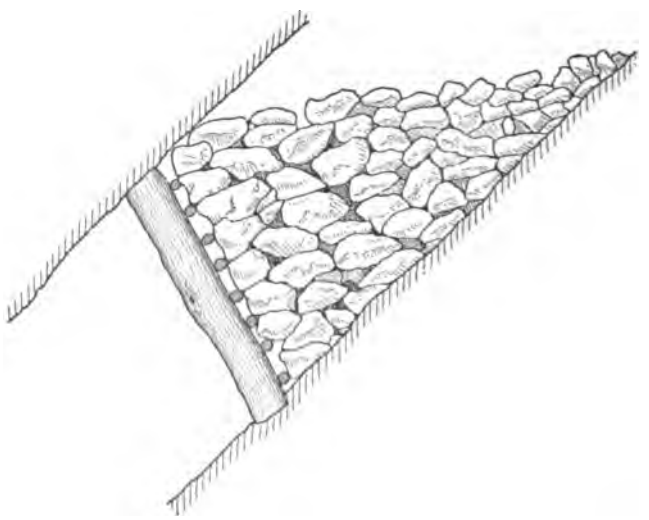


FIG. 158.—Diagram illustrating the formation of a stull.

The waste should be walled wherever possible, especially where the dip is steep, for the timber obtainable along the Rand does not last more than a few years underground, whereas walling remains good as long as it is undisturbed.

With flat reefs walling alone is generally sufficient, and in these stopes the stulls can be arranged to advance practically parallel with the stope face wherever the amount of waste is sufficient, as shown in the following diagram (Fig. 159), which

is taken from a stope on the Main Reef Leader in the eastern portion of the Geldenhuis Estate.

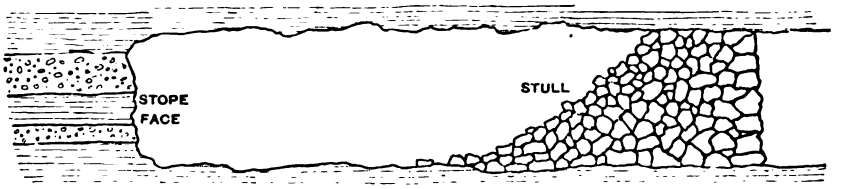


FIG. 159.—Horizontal section across a stope where the reef is dipping at a low angle, showing the formation of a stull by walling, and the relation of the stull to the working face.

Where a travelling way is made down through an old stope the weight of the roof is taken on stull pieces, regularly placed on each side, which are backed up by stulling, as shown in the following sketch (Fig. 160). Or where the reef

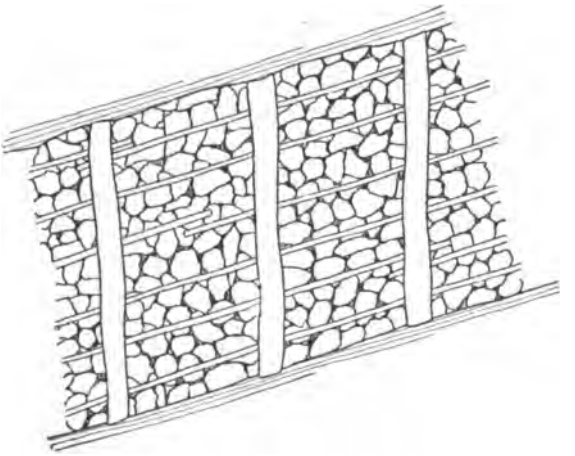


FIG. 160.—Stulling along a travelling way.

is flat, careful walling on either side is sufficient, as the stull cannot run. This is shown in the following sketch (Fig. 161).

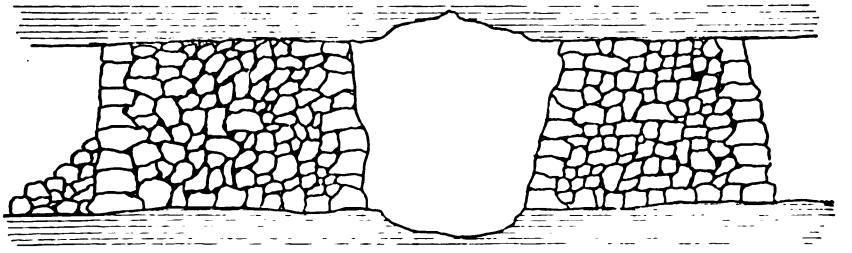


FIG. 161.—Walling of waste to protect a level on a flat reef after the ground has been worked out on either side.

Timbering.—Though timbering is sometimes used to keep the roof up in a working stope, it is in greater use to keep up small portions of the roof which hang heavily, and which are not near enough to receive support from the pillars.

In the richer mines timber is used the more frequently to support the reef, because the pillars which would have to be left contain sufficient gold to more than cover all the expenses of taking them out and putting timber in. As mentioned before, in the Durban-Roodepoort pillars are only left where the ground is very much broken, and then only when the reef is poor. In that mine the roof is practically supported on sticks of timber from 3 inches to 6 inches in diameter, which are placed about 4 feet apart. There are, in addition, pillars along both the levels, but as the distance between these is on an average 200 feet, the greater part of the roof is on timber. The stopes are not large, so that the length of timber required is only from 36 inches to 42 inches. With the kind of timber available on the Rand, were the length used in the stopes much greater than this, there would not be any great degree of security, for the sticks are often crooked, especially the longer ones.

Stope Boxes.—These are the shoots through which the broken ore at the bottom of the stope is delivered into the trucks which run along the level.

There are various kinds in use, depending upon the dip of the reef and the amount of ore which will have to pass through them.

Two drawings of a form which is in frequent use at an ordinary angle of dip are shown in Fig. 162. The box is made of pieces of deal cleated together and spiked to uprights which are fixed in the drive. Its length is about 6 feet and its width varies from 2 feet 6 inches to 3 feet. Its bottom is placed at a good slope, so that the ore will run easily along it, and it is supported at the upper end on the floor of the stope, and at the lower end on a cross-piece between the uprights; it is also further held by spikes through

the cleats on the sides. Such a box holds in itself more than a truck-load of ore, and if the box-hole through the pillar be

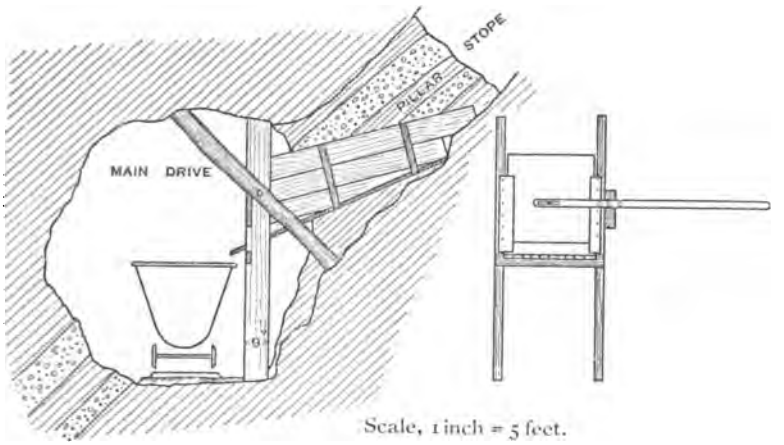


FIG. 162.—Stope box for a reef dipping at a fair angle, with drawing of box-door.

filled, the capacity is considerably increased, so that when the door is opened the discharge of the ore needs little assistance.

In the larger mines the door is an iron plate which moves between two small plates fixed on either side to the uprights. On the centre of its outside face there is a pin, which is in connection with a lever, having its fulcrum on one of the uprights, so that when the handle is lowered the door is raised.

The pin is either attached to the lever by a link, such as is shown in Fig. 163, or it runs in a slot, as shown in Fig. 162.

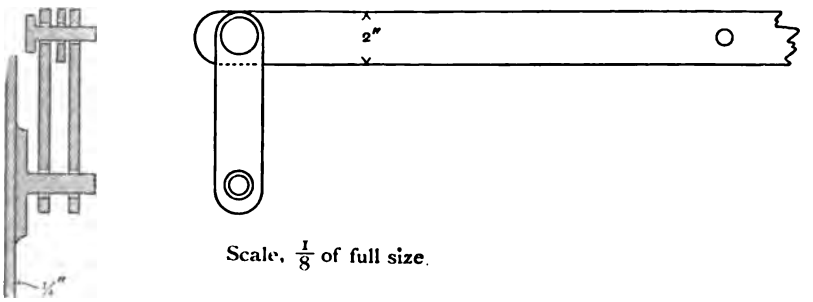


FIG. 163.—Link, lever and pin, for opening and shutting a box-door.

A door which is less costly and less substantial is made by laying pieces of deal across the mouth, to rest against

iron brackets fixed on the uprights, as shown in Fig. 164. These pieces when required are easily removed and replaced.

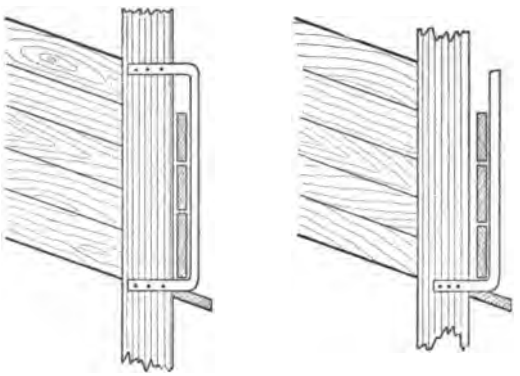
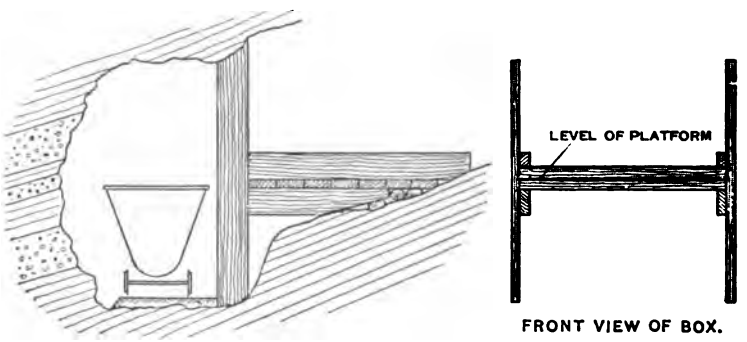


FIG. 164.—Doors for stop boxes.

Where the reef is flat, platforms are erected in the open spaces between the level pillars, from which the ore is shovelled into the truck, as shown in Fig. 165. These platforms are generally 6 feet wide and are covered with sheet-iron. Along the outside edge a board 4 inches high is placed, to prevent the ore from falling into the level. On either side of the box the openings are more or less blocked up with large waste roughly walled.

In the Durban-Roodeport mine these shoots are made by rolling iron sheets 2 feet wide into a semicircle, which is fixed on a rough support.



Scale, 1 inch = 5 feet.

FIG. 165.—Stope box for a reef dipping at a low angle.

When a box-hole has been walled up and finished with, the box is taken away to be used in another place.

Removing the Broken Ore.—The ore is generally taken to the bottom of the stope by shovelling, the amount of which

depends upon the method of stoping, upon the dip of the reef, and upon the nature of the stope floor.

With overhand stoping there is less shovelling to be done than with underhand stoping, and so, all other things being equal, an overhand stope is preferable on this account. With a steep dip there is very little shovelling necessary, as the ore falls to the boxes by gravity. At a dip of 30° and with backs of about 150 feet, 20 per cent of the boys in the stope are shovelling. In the Durban-Roodepoort, owing to the low size of the stope and the closeness of the timbering, almost half the boys in the stope are shovelling.

Where the reef is flat and the stope a large one, rails are sometimes placed in the stope. This is often done along an intermediate drive, which has become necessary by reason of the intrusion of igneous matter or the dislocation of the reef by fault action. Along such drives the trucks are run in and the broken ore is shovelled into them, the track being generally graded to the main drive so that the trucks run down easily. In the east end of the Jumpers mine, between two levels, there were 190 feet of backs, of which the top 90 feet were worked as an underhand stope and the lower 100 feet as an overhand stope. The ore from this latter was shovelled down into the boxes, but that from the top 90 feet was taken down by a self-acting incline.

Before the roof of a stope is allowed to come in, or before any stulling is done, it is usual to sweep the floor and collect all the fines which have become lodged in the holes.

Breaking the Ore in the Stope Face.—In underhand stoping the ore is broken from the face by a series of shot-holes, which are more or less put down in the direction of the dip, and in overhand stoping by a series which are only inclined at a slight angle downwards from the horizontal. The working face is always arranged in benches, in relation to the free sides of which the holes are so placed that when blasted they leave similar benches. The direction of the shot-holes and

the formation of benches are shown in the following diagram (Fig. 166).

Where there are well-developed planes of bedding which would make good walls, the holes are placed slightly inclined towards them, but not actually breaking through them, for in blasting the rock is sure to break from them if the bottom of the hole is within reasonable distance. It frequently happens that a piece of banket is "frozen into" the roof of a stope, so

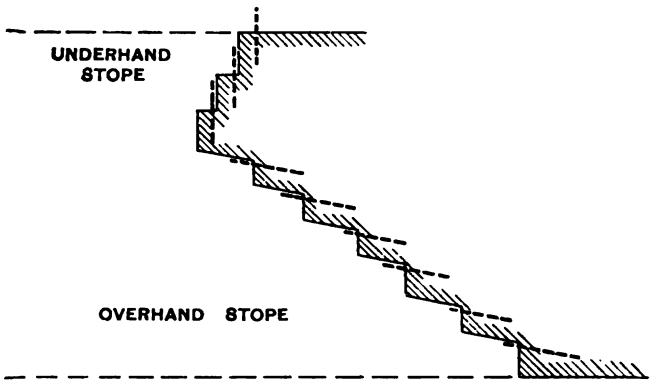


FIG. 166.—Diagram showing the direction of the holes and the formation of benches in underhand and overhand stoping.

that it has to be taken down by skimming holes all along. Such an occurrence is not, however, common in the footwall, because most of the banket is free on that side.

Where the footwall of the reef is of soft rock, as is the case in the Nigel and Blue Sky, the reef is mined by taking out the footwall first and blasting down the reef afterwards. This method was also used in the upper levels of the Robinson on the Main Reef Leader.

In a very large stope, 10 feet wide and upwards, the face is generally advanced in benches, the top one being taken first and the lower ones afterwards by a series of downright holes, as shown in the following sketch (Fig. 167).

This method is especially applicable where the reefs are flat, because the upper reefs are drilled before the lower ones are taken out. If these latter were taken first, the upper one

would be reached with difficulty, though where the reef dips steeply this difficulty is lessened. In an ordinary-sized stope generally two, and sometimes three, holes are required, more or less, across the width. Of these the lower one is generally blasted first to provide a cut for the others. With hand labour and in hard quartzite these holes are generally 3 feet deep. In smaller stopes, 3 feet wide and less, the holes are generally placed about 30 inches apart and alternately in the roof and floor, and they are generally only about 2 feet deep.

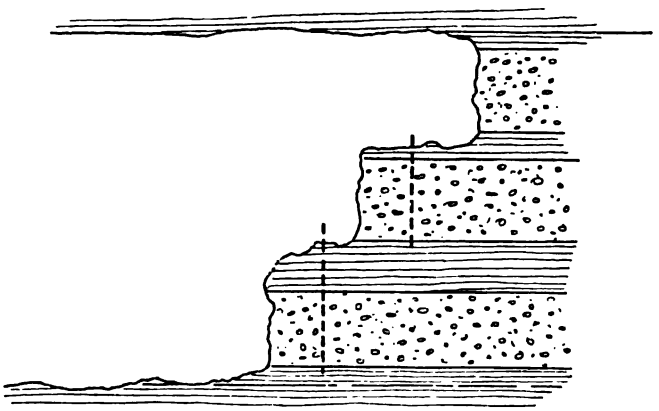


FIG. 167.—Diagram showing the method of blasting down the reef in the face of a large stope.

With hand labour the holes are drilled with $\frac{7}{8}$-inch octagon steel, and, owing to the larger size of the drilling edge of the bit, they average over an inch in diameter.

In consequence of the scarcity of native labour, machine drills, identical to those in use for development work, are to a large extent used in stoping. In large and flat stopes, with a compact reef and a good roof, they are at least as economical as hand drilling, but in small stopes, unless the walls are very good, it is hard to keep the stope down to its proper size, and in steeply inclined stopes it is not possible to so rigidly fix the bar that the arm may be used with it, in consequence of which the machine has to be clamped to the bar and its range of action is somewhat limited. In stopes where the reef is interstratified with a good deal of quartzite, some of this is so

pulverised by the large charges used in machine-drilled holes that it cannot be sorted.

In addition, where the roof of a stope is bad, more pillars are required to be left with machine drilling than with hand drilling, for the heavier charges used with the former shake up a poor roof considerably.

In addition to pulverising some portion of the rock excessively, machine stoping also brings down a number of large blocks which have to be broken up by further blasting.

In consequence of the greater amount of waste broken and of the greater proportion of unsortable waste formed, the extended use of machine drills in stoping is often accompanied by a fall in the grade of the ore milled. This is illustrated in the following results from the May Consolidated:—

Month.	Number of Machines in Stopes.	Dwts. of Bullion per Ton, from the Plates.
April 1897	10	6.49
May ,,	15	6.49
June ,,	20	6.25
July ,,	20	5.84
August ,,	20	5.29

From June to August no stoping by hand labour was done.

With a lighter class of drill it is likely that better results may be obtained.

On the score of cost, machine stoping in an ordinary-sized reef is more expensive than hand stoping. In the Crown Reef the stope contractors get 80s. per square fathom whether they use hand or machine drills, whilst with the latter the Company defrays the whole cost of the air and machines, which amounts to another 15s. or 20s. per fathom.

In the City and Suburban stopes, machine and hand contractors would get about the same money in the same stope, and in addition to being charged with native labour, native food, explosives, lights, general charges, etc., common to both contractors, the hand-drill man has also to pay, for drill-

sharpening, 5s. per fathom, and the machine-drill man for lubrication, but not for air or maintenance of the machine.

In the Jumpers mine, where four "Little Giant" drills, $2\frac{1}{4}$ inches diameter, are in the stopes, it is found that with one white man and five natives looking after two of them, stoping with these machine drills costs about as much for labour, air, and explosives as labour and explosives cost in hand drilling, so that the cost of maintaining the drills, which is considerable, has to be met over and above the costs of hand labour.

These drills are now used in several mines for stoping purposes. They are about 150 lbs. in weight, and they run with less than half the air necessary for the larger drills of $3\frac{1}{4}$ inches diameter and with $1\frac{1}{2}$ inch less stroke. It is found that ordinary drill steel of $\frac{7}{8}$-inch diameter can be used with them without being buckled or unduly broken. By each of these drills close upon 30 feet are drilled and 6 tons of ore broken per shift on an average, which is about the work of nine hammer-boys, who drill 3 feet and break 12 cwts. per shift. The cost of running these drills is given on p. 383.

It is especially in the matter of explosives that machine drilling is the more expensive method. Mr. Johns, in his evidence before the "Industrial Commission," gave the following information :—

"In stoping 717 square fathoms of reef by hand labour the average cost of gelatine per square fathom was 17s. 1.3d., and in stoping 899 square fathoms of reef by air drills the average cost of gelatine per square fathom was 29s. 8.6d., or 1.63 times the quantity used in hand labour."

Explosives.—In the stopes, blasting gelatine, stated to contain 93 per cent of nitro-glycerine, is used more than any other explosive, though a considerable quantity of what is called No. 1 dynamite, containing 75 per cent of nitro-glycerine, is also used; both are made into "sticks" of $\frac{7}{8}$-inch diameter.

For driving and general development work, gelatine is invariably used, and in the stopes its use is being further

extended. Where, however, the stopes are not on contract it is advisable to keep to the use of dynamite in them, for if gelatine be used, it is likely that the miners contracting in the drives will get some of it.

The amount of gelatine used per ton of ore broken in the stopes depends upon the size of the stope, upon the percentage broken by machine drills, and upon the nature of the rock. In the Langlaagte Estate, where the average size of the stopes is more or less 8 feet, about ½ lb. of gelatine is used per ton of unoxidised ore broken, and in the Princess Estate, where the stope is about 3 feet wide, about 1¼ lb. of gelatine is required.

From the figures given previously, it can be calculated that in the Ferreira mine, with hand drilling, 0.60 lb. of gelatine is used per ton of ore broken, and with machine drilling 0.98 lb., in stopes which average 4½ feet in width.

Blasting in the stopes is done once each shift, except in those mines which are working in red ore, and in those other mines where a native puts in two small holes per shift instead of one deeper one.

Labour in the Stopes.—In those mines where the reef lies regularly and at a fair angle of dip, it is usual to let the stopes out on contract. The unit in general use is the square fathom of area on the reef plane; with this system no payment is made on the amount of ore put out by the contractor, so there is no inducement for him to mine waste; the inducement is to carry the stope at the best width for breaking, independent of the presence of reef, but this can always be guarded against. The tendency with this method of payment and with reefs of an average size, say 4 to 5 feet, is to get as clean a product as is possible.

The contractor employing hand drilling is charged with native labour, native food, explosives, lights, timber charges, etc., and in some cases with drill-sharpening. A contractor using a machine drill is generally charged with the above

items, with the exception, perhaps, of drill-sharpening, and in addition he has to pay for the lubrication of his machine. On these terms, prices for contracting with hand labour are practically the same as with machine drills.

When a pillar has to be left, the miner is allowed something for cutting round it; similarly, he is generally allowed something extra for putting a box-hole through to the level, but the extent of these considerations depends upon the original prices.

In order to encourage good work a bonus is given in some mines to the contractor who gets his rock out with the least expense, and also to the one who fills the greatest number of trucks with reef.

It is common practice, where two miners are partners in a contract, that they each have their own stope face to work, the two stopes being as near together as possible. It is then arranged that as each man goes down to work, he puts a certain number of his boys to clean up his partner's stope; the advantage of this is that each man has a clean stope face upon which to start drilling as soon as he gets down, and the object of having separate stopes is that their respective rates of progress may be compared.

In most mines the hammer-boys understand that they have to put down a hole 3 feet deep per shift, or if this hole be an upper or a dry hole, a smaller length is required. In the red ore the ground is so soft that two holes are always put in, and sometimes three, and this is also the case in the Nigel mine, where the slate footwall of the reef is soft.

In the Roodepoort district, where the stopes are small, and where a deep hole would not be advantageous, a native is required to put in two holes, each 2 feet deep, and blasting in the stopes is done twice each shift. With a hole 3 feet deep on an average-sized reef, it is estimated that 12 cwts. of ore are broken.

The shovel-boys are supposed to keep at their work till it is finished or till blasting commences again.

In order to compare the amounts of work which are being done in different stopes, records of the numbers of trucks of ore obtained per hammer-boy and per shovel-boy on each shift are often of service, and these should vary but little for all the stopes in the same mine, and if there is any variation the reason should be obvious. These coefficients are of great use to the mine manager.

There are few tributors working along the Rand; in one

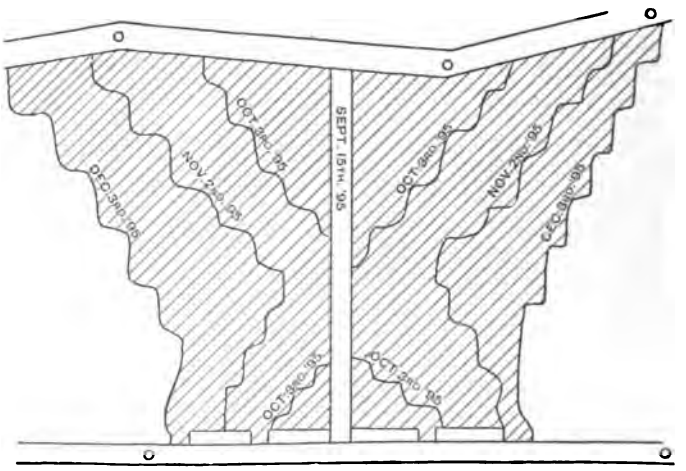


FIG. 168.—Diagram showing method of keeping up a stope plan.

or two instances entire mines have been taken on tribute, and generally the tributors did well. Some years ago in the Robinson, portions of the South Reef, where it consisted of leaders separated by considerable thicknesses of quartzite, were let out to tributors, who were paid by the assay value of the ore they brought to the mill.

Stope Plans and Ore Account.—The stopes are surveyed in the larger mines at the end of every month, and in the smaller mines at longer intervals. These surveys are in most mines plotted on the working plan. The full advantage of stope surveys is, however, obtained when, in conjunction

with samples, they are plotted on a projection on the reef plane, as mentioned in the chapter on sampling. In one or two cases the stope plans are made on the reef plane, as, for instance, at the Crown Reef, and it is from these plottings by means of a planimeter that the areas worked out are obtained. From these figures, and knowing the width and specific gravity of the reef, the number of tons which have been mined from any one stope can be determined, such determination being more accurate than that obtained by counting the number of trucks which have been filled from any stope, for in a large mine as many as sixty stopes are sometimes working, and the work of tallying the trucks becomes very difficult.

The preceding diagram (Fig. 168) shows the method of keeping stope plans.

Every stope face as it is plotted is marked by a fine line, along which the date of the survey is placed, and the "hatching," which indicates ground worked out, is brought up to cover the last area.

Where the assays are indicated in the same plan, the area worked out is better shown by a wash of colour.

CHAPTER XVI

AIR-COMPRESSORS, ROCK-DRILLS, ETC.

Air-Compressors.—The compressors along the Rand are, with one or two exceptions, direct steam-driven and not power-driven; that is, the air and steam cylinders are rigidly connected to the same piston-rod, as shown in Figs. 169, 170. With the exception only of the Willans compressor, the air-cylinders in use are double-acting, air being compressed with each stroke of the piston.

The majority of compressors are horizontal; the style which has up to the present been chiefly used is known as the "Duplex"; this is characterised by consisting of two halves, each in itself a complete compressor, one right-handed and the other left-handed, with a heavy fly-wheel between the two, running on a shaft on to which the respective cranks are keyed at right angles to one another. This type of compressor is illustrated in Fig. 169; one half of the machine can be erected first, and later on, should it be necessary, the other half may be added, thus doubling the capacity.

If for any reason one half of a duplex compressor only be required, the other half is easily disconnected, so that, while one half is undergoing repairs, the other can be in use.

In these the air is compressed in one stage, though a tandem compound duplex might be introduced with advantage.

It is now more usual to compress in two stages, in much the same way that steam is expanded in two cylinders, and

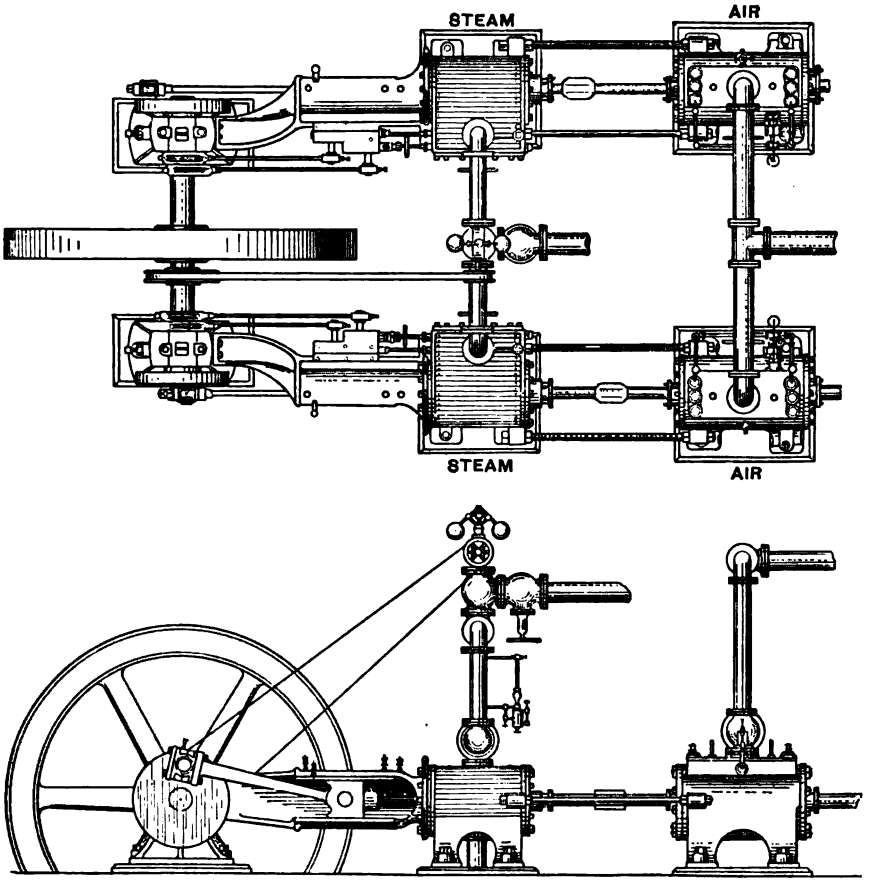


FIG. 169.—Duplex compressor, single stage (Ingersoll Sergeant).

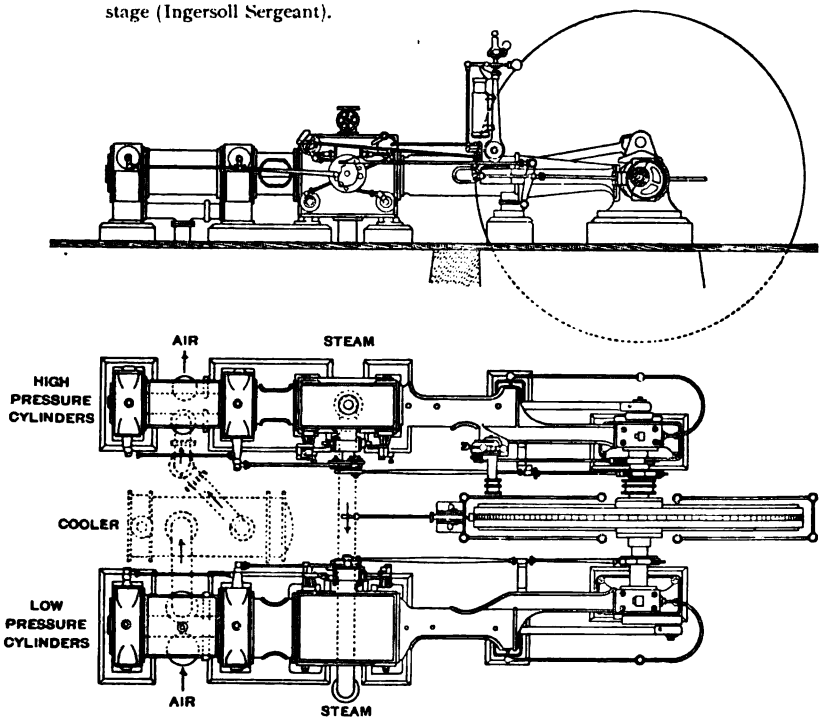


FIG. 170.—Cross compound compressor, double stage, Fraser and Chalmers (Riedler system).

with a similar gain in economy. This is done by admitting the air into the first and larger cylinder, where it is compressed to about 20 lbs. per square inch above atmospheric pressure, and then delivering it into an intermediate cooler, to reduce the temperature caused by compression, before passing it into the second and smaller cylinder, where it is finally compressed.

By double-stage compression in this manner, an amount of air can be compressed with an expenditure of power appreciably less than would be required for single compression. This is shown in the following tabulated statement of results, obtained by Mr. K. Schweder, and as given by him in his paper, "The Relative Mechanical Efficiency of Air Drills and Electric Drills," read before the South African Association of Engineers and Architects, 24th November 1897.

Investigated Plants.	Cubic Feet of Air compressed to 65 lbs. per minute per I.H.P. at the Engine.	Revolutions necessary to make up for the Losses in Compressor and Main Pipe.	Loss in the Main and Compressor as Percentage when all Compressors were running 80 Revolutions.	Loss in Cubic Feet per minute as Leakage per 1000 Square Feet Surface of the Pipage.	Cubic Feet Compressed Air delivered to each Drill per minute.	Indicated H.P. at Engine per Drill running.
1. Langlaagte Estate—Rand Drill . .	5.35	13.2	16.5	27.8[1]	58	13.7
2. Crown Reef—Ingersoll Sergeant	5.65	37	46.25	20	77	24.5
3. Meyer and Charlton—Hirnant Drill Co. .	5.25	14.3	17.8	110[1]	46	13.5
4. George Goch—E. P. Allis . .	5.85	6.8	8.5	26[1]	98	19.5
Average . .	5.52	17.3	22.3	58.4	70	17.8

[1] Includes the slight leakages in the compressor.

Nos. 2 and 4 are double-stage compressors, and it is seen that they compress a greater volume of air per indicated

horse power than do the others, which are single-stage compressors.

According to Mr. C. T. Roberts (*Proceedings of the South African Association of Engineers and Architects*, vol. i. pp. 14, 80) a single-stage compressor at the Metropolitan was replaced by a compound compressor, with the result that the latter drove 20 drills with a consumption of 163.8 I.H.P., as against 197.4 I.H.P. absorbed by the former.

The gain in economy is generally estimated to be from 10 to 15 per cent; it follows upon the relatively smaller volume of colder air, which allows the cylinder to contain more of it. With this fact in view, the intake for the air to be compressed is always arranged to be in as cool a place as possible.

The loss of capacity owing to the necessary clearance between the cylinder-heads and the piston at the end of each stroke is not important; it is practically remedied by making the cylinder a little larger, because the work performed in compressing the air into the clearance space is to a great extent given out during the succeeding stroke. The usual amount of clearance is about 3 per cent of the cylinder volume; air compressed into this at 75 lbs. pressure expands as the piston recedes, and air is taken in until it reaches about 15 per cent of the cylinder volume, so that, with single compression to 75 lbs. pressure, the capacity of the cylinder is reduced 15 per cent by the clearance.

With double compression, in the larger cylinder, owing to the smaller pressure reached, the effect of a 3 per cent clearance is not much more than to reduce the capacity of the air cylinder by about 5 per cent.

In consequence of the rarity of the air at the altitude of Johannesburg (about 5650 feet above sea-level), the capacity of the compressors is lessened; it is usual to expect them to perform about 84 per cent of what they will do at the level of the sea.

The type which is now being extensively erected is the

horizontal cross compound, such as is illustrated in Fig. 170. With these, the high-pressure cylinders, steam and air, are in line on one side of the fly-wheel, and the low-pressure cylinders are on the other side, connection between the respective cylinders being made across the space between them. With these engines it is often arranged that the high-pressure half may be erected first, to be compounded afterwards, should it

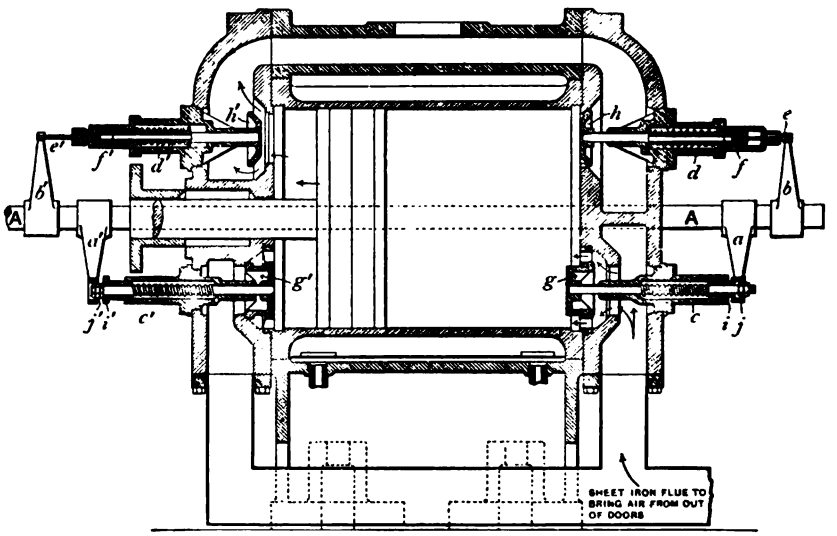


FIG. 171.—Air cylinder with poppet valves, Rand Drill Company.

be required, but each half, when both are erected, is not made to run independently of the other, as with the "Duplex."

There are several vertical compressors in use which, except in respect of their upright position, are identical with those that are horizontal. Though this type costs more initially, with very large engines it is less expensive in wear and tear and loss in friction than the horizontal type. There are two or three vertical compressors with triple expansion steam cylinders; with these there are two low-pressure and one high-pressure air cylinders.

The different makes of compressors differ chiefly in the construction of the air valves.

In Fig. 171 the air cylinder of the Rand Drill Company

of New York is shown; it is arranged that the springs at the back of the poppet valves are withdrawn just when the valve should open, so that a full opening is afforded whilst the air is passing out, after which the spring comes back into position to close them.

In Fig. 172 the inlet and outlet valves of the Riedler compressor of Fraser, Chalmers, and Company are shown.

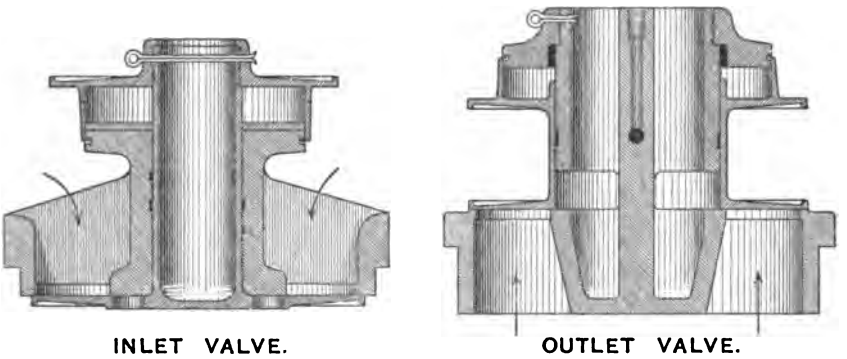


FIG. 172.—Riedler air valves.

These valves are opened by air-pressure against resistance provided by a dash-pot, and closed by a finger worked from the wrist plate of the Corliss gear, the extreme movement being about $\frac{7}{8}$ inch.

The piston inlet valves and air cylinder of the Ingersoll Sergeant Company are shown in Fig. 173; the valves are annular, one on each face of the piston, the interior of which is in connection with the outside air through a hollow tubular continuation of the piston through the back head of the air cylinder. The extreme movement of these valves is about $\frac{1}{4}$ inch. The outlets are provided with spring poppet valves.

All the compressors used are "dry" compressors, that is to say, any water used for cooling the air does not come in contact with the air itself, but, instead, the air cylinders are as far as possible covered with jackets, through which cold water circulates, as indicated in Fig. 173. The arrangement of the

valves in the Ingersoll cylinder allows the cylinder-heads also to be water-jacketed.

With double-stage compression there is between the two air cylinders an intermediate cooler, into which the air from the low-pressure cylinder is received, to be cooled before entering the high-pressure cylinder.

The size of a compressor depends upon the number of drills it will have to run. As shown by Mr. Schweder in his

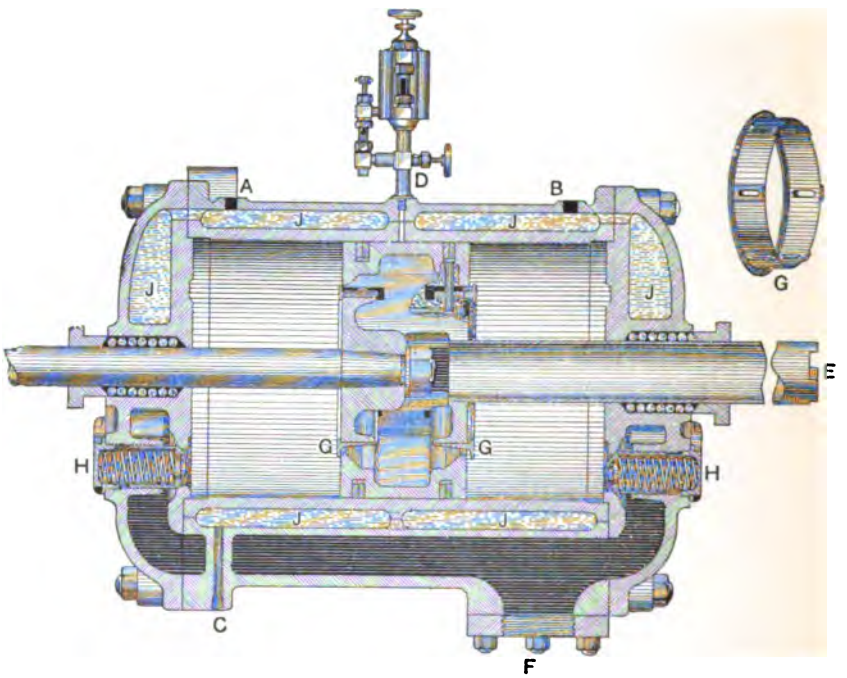


FIG. 173.—Ingersoll Sergeant piston inlet air cylinder.

table, page 368, with the four compressors experimented upon, the average volume of compressed air delivered to each drill per minute was 70 cubic feet, though the various volumes were greatly different; the actual amount of air which is received at the drill is of course less than this, owing to the loss by leakage, as stated on p. 368.

The usual pressure of the air delivered by the compressors is between 70 and 80 lbs. per square inch above that of the

atmosphere. At the time of the tests by Mr. Schweder, the Meyer and Charlton compressor was running at 55 lbs. pressure, the Crown Reef at 75 lbs., and the George Goch at from 83 to 88 lbs. per square inch, though they were all compared at 65 lbs.

The maximum speed at which compressors are run varies from 65 to 100 revolutions per minute; the smaller ones as a rule run at a higher rate than the larger ones. The speed is generally regulated in accordance with the amount of compressed air required, by a governor especially constructed to act directly on the cut off, which in most cases has a range of from nothing up to 70 per cent of the stroke, the throttle valve remaining fully open. The ordinary speed governor is only arranged to act should the engine, from breakage of the mains or other cause, reach the limit of speed.

Receivers.—After having been compressed, the air passes into a receiver of large capacity, so that the intermittent delivery from the compressor is rendered much more uniform before the air passes into the mains. These receivers are usually cylinders of sheet-steel from 4 to 5 feet in diameter and 10 feet to 20 feet long; they are fitted with safety-valves, pressure gauge, blow-off cock, test cock, etc. Should there be any water vapour in the air, it is condensed in these cylinders and can be drawn off. In the Ingersoll compressor the escape from the safety-valves of the receiver is led back to cut off the steam and balance the pressure of air on either side of the air piston, so that no further compressing is done until the pressure in the receiver is relieved.

Air Mains and Pipes.—The pipes which are used for the conveyance of compressed air underground are generally lap-welded iron or steel tubes; wrought-iron pipes are also used, and, for small diameters, butt-welded pipes.

The usual diameter for air mains down the shaft of the larger mines is about 6 inches. The diameter depends upon

the amount of air which has to be supplied; the smaller the pipe the greater the velocity of the air in it and the higher the friction against the walls of the pipe. It has been demonstrated that with an air velocity of from 25 to 30 feet per second, the friction only amounts to 2 lbs. per square inch per mile of main; generally the velocity should not exceed 50 feet per second.

Along the levels smaller pipes are used, and generally each drill is connected by a length of hose with a pipe 1 inch in diameter.

The usual form of joint is shown in Fig. 174; the flanges, which are of cast iron, are loose and in halves, so that they

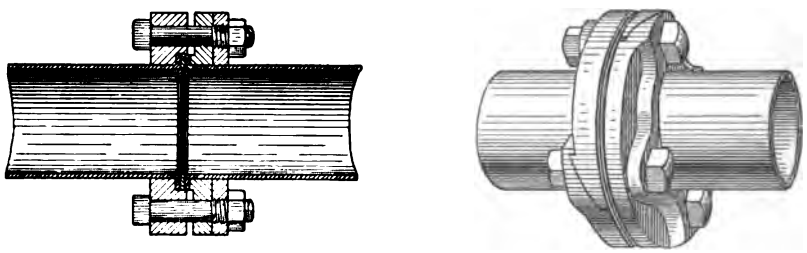


FIG. 174.—Stewart's patent flange joint.

may be easily removed. At the joints the end of each pipe is turned up to make a "beading" about $\frac{3}{8}$ inch deep; the tubes are then butted together, with a rubber ring, or gasket, between them, and the joint is made by two of Stewart's patent flanges, one of which is relatively male and the other female, as shown in the figure. These two flanges are bolted together with four bolts, and everything is brought up tight. Sometimes one flange is loose and the other tight, or they may both be made to screw on. For the purpose of turning the line of pipes through a small angle, oval flanges are sometimes used. The joints are sometimes made by screw couplings instead of by flanges, as shown in Fig. 175. This joint can be used for very high pressures such as are, however, not found in air mains underground.

Whatever the joint, the leakage in air mains is con-

siderable, as shown in Mr. Schweder's table on p. 368, in which the average loss of air per 1000 square feet of pipe surface for the four air services mentioned is stated to have

FIG. 175.—Screw coupling joint.

been 58.4 cubic feet per minute. This figure was obtained by running the compressors into the air mains with all the drill cocks closed.

Machine Drills.—The machine drills used all belong to the type known as percussive or reciprocating drills, the drill steel being practically an extension of a piston-rod. The piston is actuated by compressed air, and works in a cylinder which slides in a shell or guide, so that it can be advanced towards the face as the hole is deepened.

This feed motion is effected by a screw, and in an ordinary-sized drill of about $3\frac{1}{8}$ inches diameter its length is about 24 inches.

The rotation of the bit is effected by the movement of a rifle nut, forming part of the rear end of the piston, along a rifle bar which extends into the piston through the rifle nut. This rifle bar is so held by a ratchet and pawl, that it causes rotation of the drill during the backward stroke, and is itself caused to rotate during the forward stroke, so that the drill only turns in one direction and only on the back stroke.

The drills used along the Witwatersrand may be divided into two main groups according to whether the air is distributed by a valve moved by the pressure of the air, as shown in Figs. 175, 176, 177, or by a valve thrown by some positive movement directly transferred from the piston, as shown in Fig. 178. These two valves are generally spoken of as "pressure valves" and "tappet valves."

The pressure valves are in greater use, chiefly because

there are less breakages with them, and also because with tappet valves and with a long stroke, the air is more likely

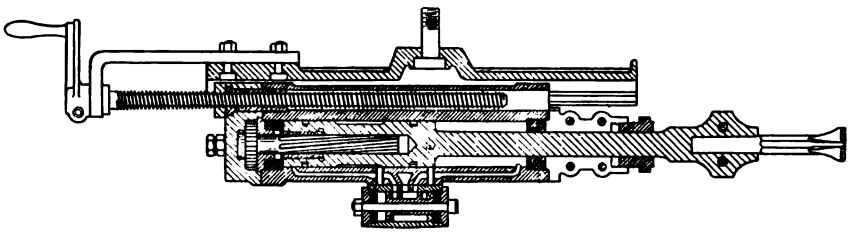


FIG. 176.—Ingersoll Eclipse drill.

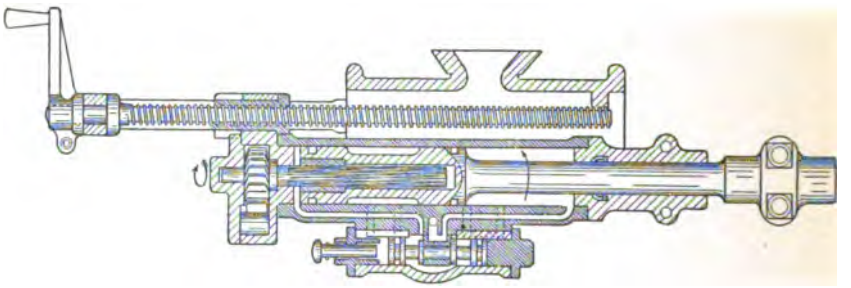


FIG. 177.—Climax air valve drill.

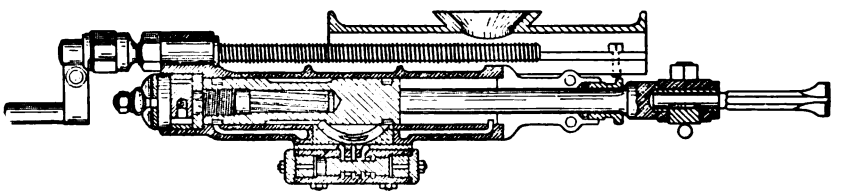


FIG. 178.—Ingersoll Sergeant drill.

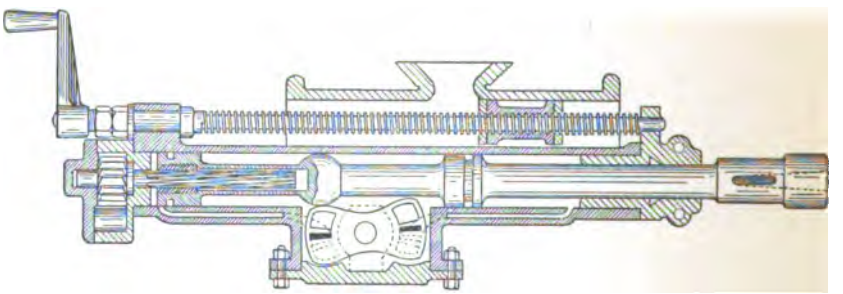


FIG. 179.—Climax tappet drill.

to be let in to reverse the movement before the end of the stroke, thereby causing more or less cushioning.

The following are the drills which are chiefly used:—

The Slugger.—With this machine the valve is moved by pressure, being actuated by differences of air-pressure brought about during the travel of the piston. The valve consists of one piece of metal in the shape of a small piston, having at the utmost a movement of $\frac{3}{4}$ inch; it is made of hardened steel and grooved to fit its cylindrical chamber. These machines are often constructed to use the air expansively; they are made by the Rand Drill Company of New York, and the size called No. 13, of which particulars are given in a following table, is that most frequently used.

Ingersoll Eclipse.—This drill, illustrated in Fig. 176, is also worked with a pressure-thrown valve. The parts are so arranged that the piston is, within certain limits, capable of working with a variable stroke. In starting a hole, a shorter stroke, such as could be obtained with this machine, is better than a longer one. This drill is made by the Ingersoll Sergeant Company of New York; particulars of the size which is most frequently in use are given in a following table.

Climax.—This drill is shown in Fig. 177. In it, as in all other pressure-thrown valves, the valve is a spool, with piston ends working in a cylindrical chamber; the movement is effected by the transference of air pressure from one end of the valve to the other, at the uncovering and covering of proper ports by the piston in its travel. This drill is made by R. Stephens & Son, Cornwall. These makers also make a tappet drill, shown in Fig. 179.

Ingersoll Sergeant.—In this drill the main valve, which distributes the air, is an air-thrown valve, similar to those already mentioned, but instead of being actuated by the covering and uncovering of ports, it is worked by a movement transferred from the piston to a tappet as shown in Fig. 178. This drill is sometimes considered a better drill than the Ingersoll Eclipse for very hard rock, but in the ordinary

ground on the Witwatersrand the Eclipse is preferred to it, as it works with less air.

In the following table some particulars of these drills are given :—

	Slugger No. 13.	Ingersoll Eclipse.	Climax.	Ingersoll Sergeant.
Diameter of cylinder	3⅛ inches	3⅛ inches	3½ inches	3¼ inches
Length of stroke	6½ "	6 "	6½ "	6½ "
Length of feed	24 "	24 "	24 "	24 "
Usual depth of hole drilled	from 1 to 15 feet	14 feet	16 feet	14 feet
Usual size of bottom of hole	1¾ inch	1¼ inch	1½ inch	1¾ inch
Diameter of inlet hose	1 "	1 "	1 "	1 "
" of steel used	1⅛ "	1⅛ "	1⅛ "	1⅛ "
Weight of machine unmounted	265 lbs.	273 lbs.	290 lbs.	246 lbs.
Number of strokes, with 60 lbs. pressure at drill	...	325	400	325
Approximate weight of blow delivered on the rock at each stroke	...	625 lbs.	...	650 lbs.

For stoping purposes smaller drills, about 2¼ inches in diameter and weighing about 175 lbs. or less, have proved to be better adapted than larger ones, as mentioned on p. 361. It is likely that their use will soon be considerably extended.

In addition to the drills thus briefly described, there are many others working, including the Banket, Hercules, Holman, Hirnant, Climax, Tappet, etc. In these the Tappet type predominates.

The guides or cradle, along which the cylinder slides, are mounted upon columns, bars or tripods, according to the work to be done.

Columns are used in ordinary headings; they are made of stout iron tube about 4½ inches in diameter and 6 feet in length. The length, however, depends upon the height of the heading. At the top end there is a cap piece of larger diameter, which is pressed against a block of wood on the roof, by screw

columns or jacks beneath, as shown in Fig. 180. Along the column slides an arm of iron tube, somewhat smaller in diameter, which can be fixed at any height and in any direction, and to this arm a clamp, to which the machine drill is rigidly attached, can be secured at any point.

Single screw columns, as shown in Fig. 180, are used in small drifts and in stopes. In these latter, where the inclination is about 45°, the arm is sometimes not used, because a rigid position for the column is difficult to obtain, and the attachment of the arm makes it still more difficult.

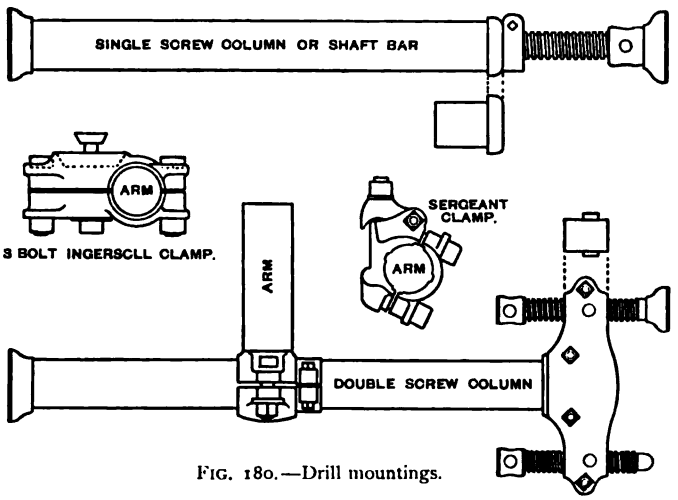


FIG. 180.—Drill mountings.

The frontispiece is a reproduction from a photograph showing a rock-drill at work in a stope, and illustrates well the manner of setting up the machine.

In shaft-sinking the single screw columns are used; they are placed horizontally from one side of the shaft to the other, and the arms are used with them.

In main cross-cuts or large drives, a horizontal bar is often used, because on it two drills can be more easily accommodated than on one vertical column.

Tripods are used where ore bins and stations are being cut out, and in other places where columns or bars cannot be rigged up.

On the Witwatersrand recently, attention has been paid to some experimental results obtained from an electric drill called the Bladray. Briefly described, this drill is a motor electric drill, working by percussion. Its general appearance, its connection with the column, its arrangements for feed and for the necessary rotation of the drill, are similar to those of the air drills now in use, and the shank to which the drill is attached at one end, and to which the percussive force is applied at the other, plays exactly the same part as does the piston of air drills.

The valve-box, of course, is absent, and in its place is the motor chest, which is arranged, with its armature inside, concentrically around the central axis of the machine. The armature is a hollow cylinder, through which the shank passes freely. To this armature, on the side towards the rear of the machine, is attached concentrically a cylindrical shell, with its unattached end arranged as a spiral cam or screw surface (No. 1). Fixed rigidly to the shank is a similar cam surface (No. 2), which does not turn with No. 1, but which is kept pressed against it by a strong spring behind the shank end.

Upon the rotation of No. 1, No. 2 is shifted back, and with it the shank, till No. 1 has reached the limit of its throw. In being shifted back it compresses a spring, which, when the point of release is reached, comes into force and strikes the blow.

In the following statement some particulars of this drill are given :—

	Large drill.
Length of stroke	3 inches
Weight unmounted but with cradle	225 lbs.
Strokes per minute	700
Approximate weight of blow delivered on rock at each stroke	400 lbs.

It is seen that this drill is to run shorter, lighter, but quicker than the air-driven drills, and it is claimed that this stroke will give a better duty than the former.

Drill Steel and Bits.—With machine drills "cross" or "star" bits and "chisel" bits are used. In drilling a hole a short drill of larger diameter is first used, and when this has drilled to its full depth, a longer one, of smaller diameter, is substituted, and so on.

The usual depth of a hole is about 6 feet. This is drilled much in the way indicated by the following figures:—

								Inches.
Cross bits	3	inches	diameter,	depth	drilled	.	.	12
"	$2\frac{1}{4}$	"	"	"		.	.	15
Chisel bits	$1\frac{5}{8}$	"	"	"		.	.	18
"	$1\frac{3}{8}$	"	"	"		.	.	27
								72

The larger diameter is made at the start in order that the drill may run easily when it is drilling near the bottom; it also lessens the chance of sticking.

The cross bits are well suited to the uniform nature of the quartzite, and compared with other forms of percussion bits they are easily sharpened. For these bits the steel is usually supplied in lengths of cruciform section, which are welded to short lengths of round iron, about $1\frac{1}{8}$ inch in diameter, on each of which, at the other end, a shank has been turned or forged. The chisel bits are made from $1\frac{1}{8}$-inch round steel; on one end they have the bit, and on the other the proper shank. With each machine drill a full complement of seventy bits of all sizes is allowed; some of these, at any one time, are sharp, and others are dulled. It is usual to find the rock is drilled away at such a rate that the mud can all be flung out by the machine with the aid of water which is thrown into the hole.

For hand drilling the steel is octagonal in section, and $\frac{7}{8}$ of an inch in diameter.

For sharpening drills several machines have been introduced. The Burness, Bradbury, and Myers Chisel Bit Machines have all been designed and used for sharpening

chisel bits, whilst the Myers Star Bit Machine and the Liddel have been used for star bits. These machines are generally power-driven.

In the report, dated 31st March 1896, of the General Manager of the Geldenhuis Estate, it is stated that the Burness machine enables one man with four boys to sharpen 2000 to 2500 hand drills (chisel bits) in ten hours.

The usual number which one blacksmith would sharpen in the same time is about 400.

Machine sharpeners have generally given satisfaction, but they have not yet been unanimously adopted. In the Robinson and other mines hand labour is still entirely used.

A contrivance for heating the drills, called "Blakneys Furnace," has lately been introduced, which possesses almost every advantage over the ordinary open forge.

Rates and Costs.—Mr. L. I. Seymour in his paper "Rock-Drilling; Comparative Results," read before the South African Association of Engineers and Architects, 29th September 1897, stated that the average aggregate depth of hole drilled per shift of 12 hours, by one machine, is from 20 to 24 feet, with a 3¼-inch drill, and he gives the following estimate of the cost of running such a drill during both night and day shifts for a month :—

	Cost per drill per month.	Per cent of total cost.
Cost of coal	£15 0 0	12.53
Oil, waste and general stores . . .	2 1 9	1.74
Repairs to drill, hoses, etc. . . .	1 10 0	1.25
Spare parts of drill, including hoses .	9 12 4	8.03
Repairs on compressor, boilers, etc. .	0 15 0	0.63
Two Kaffir stokers at £3 : 10s., and one white stoker at £22 : 10s. . .	1 0 0	0.84
Three engine-drivers, 28 shifts each per month at 18s. 4d. (take one-half of this amount as these men drive two other engines), equals £77 : 2 : 10 .	1 5 8	1.07
Sharpening bits, and cost of steel . .	12 10 0	10.43
Carry forward . .	£43 14 9	36.52

	Cost per drill per month.	Per cent of total cost.
Brought forward . .	£43 14 9	36.52
Two white men operating the drill, 28 shifts at £1 each	56 0 0	46.77
Four Kaffirs at £3 : 10s. each . .	14 0 0	11.69
Interest at 7 per cent on half of the equipment, at £450 per drill . .	1 6 3	1.10
Redemption at 12½ per cent on £450 .	4 13 9	3.92
	£119 14 9	100.00

Total cost per drill per month, say £120.

At the George Goch the figure actually obtained was £115 : 8 : 1.

With 2¼-inch drills better results are obtained; chisel bits are used throughout, the hole starting with a diameter of 1¾ inch and finishing with one of 1 inch, and greater footage is attained. The following is Mr. Seymour's estimate of running two 2¼-inch drills during both night and day shifts for a month, with one white man each shift looking after the two drills, assisted by five boys, two to each drill and one to carry water:—

Cost of coal for the two drills	£15 0 0
Oil, waste, sundry stores	3 0 0
Drill spares	12 0 0
Repairs to compressor, boiler, etc.	0 15 0
Stokers	1 0 0
Engine-drivers	1 5 8
Sharpening bits and steel	12 10 0
Drill-men	63 0 0
Kaffirs	35 0 0
Interest on equipment	1 6 3
Redemption	4 13 9
Total cost per month for running two 2¼-inch drills	£149 10 8

At the Jumpers mine machines of this size were each doing over 25 feet per shift, which works out that drilling with these machines costs about 1s. 2d. per foot drilled, against 1s. per foot for hand labour. With the larger drills the cost is very much greater. Air drills are usually run at an average

pressure of 60 lbs. per square inch above atmosphere. At this pressure they deliver about 350 blows per minute with a stroke of 6 inches, and, as stated by Mr. Seymour, drill about four holes, in all 20 to 24 feet, per shift.

At the George Goch the drills are run at a pressure of from 83 to 88 lbs., and they hole about 36 feet, in six holes of 6 feet each, per shift. As the results of indicator diagrams taken from an air drill, and of some experiments and observations upon the number of foot pounds in the piston and drill at the time of striking, Mr. Schweder has determined that the mechanical efficiency of the drill used was 87 per cent.

Mr. Schweder also found that the loss by leakage from the compressor to the drill, and for filling the clearances of the drill, is about 42.7 per cent, or the efficiency of the transmission is $100 - 42.7 = 57.3$ per cent.

The particular experiments which he made to determine the mechanical efficiency of the whole air-drilling equipment were at the George Goch, where the following figures were obtained: "An air drill giving 405 blows, $5\frac{1}{8}$-inch stroke, gave for the down stroke alone 1.95 I.H.P. and 1.7 E.H.P. (actual horse power exerted), thus: mechanical efficiency, $1.7 \div 1.95 = 85$ per cent; the drill took, on the double stroke (up and down), according to indicator diagram on the drill, 49.5 cubic feet per minute, and, according to the diagram from the engine, 98 cubic feet; consequently loss and leakage and filling the clearance, $98 - 49.5 \div 98 = 49$ per cent." "The total mechanical efficiency of the former air drill equals $1.7 \div 17.8$ or 9.55 per cent."

The figure 17.8 is the average number of I.H.P. consumed per drill running, in the four plants experimented upon, as stated in the table on p. 368. As the compressor of the Crown Reef was in an abnormal and remediable state of leakage, it is better to use the figures of the other three, which were being run in a fair condition; these give with 15.5 I.H.P. per drill an efficiency of 11 per cent. It is given

below that 13.5 I.H.P. was consumed per drill on the Jumpers Deep, but this figure is not on a basis for comparison with those of Mr. Schweder, because those were obtained whilst a known number of drills were continuously pounding away at the rock. Accepting this figure, 11 per cent, for total mechanical efficiency, it would appear that the efficiency of the conversion from the indicated power on the engine, to compressed air delivered, and again the conversion into work of the compressed air received at the drill must be very small.

The efficiency of the first conversion with high-class compressors is generally estimated to be somewhat over 80 per cent, so that that of the second must be about 30 per cent. A compound machine drill of the Hirnant type shows, by indicator cards, a saving in air of nearly 40 per cent as compared with the ordinary machine, but it drills at a slower rate.

The following information relative to the rock-drills running at the Jumpers Deep was kindly given me by the manager, Mr. Strangman Hancock.

DETAILS OF RISING, DRIVING, AND SINKING DONE

	Size of Excavation.	No. of Drills engaged.	No. of Feet advanced.	No. of Cubic Feet cut.
	Feet.			
Sinking vertical shaft .	8 × 23	4	441	81,144
Cutting underlay „ .	8 × 23	4	30	5,520
„ stations . .	21 × 12	2	60	15,120
„ ore bins . .	20 × 18	2	65	23,400
Driving main cross-cuts	10 × 8	2	1287	102,960
Prospecting „	7 × 6	1	243	10,206
Double drives on reef .	10 × 8	2	874	69,920
Single „ „ .	7 × 6	1	1751	73,542
Winzes and rises .	6 × 4	1	930	22,320
		Totals	5681	404,132

Average indicated horse power for 20 drills, 270; average indicated horse power per drill, 13.5; average number of revolutions of the compressor per minute, 35.453.

Summary of Duty

Total feet driven, risen, sunk	5,681
Total cubic feet cut out	404,132
Average number of cubic feet of rock section per foot driven	71
Average footage per month per drill on above section	40
Average number of cubic feet cut out per drill per month	2,881
Equivalent of each foot cut out in tons	5·47

Converted Duty on each Drill per Month on Normal Section,
i.e. 7 *feet* × 6 *feet*

Equivalent number of feet driven	9,607
Converted cubic measurement of rock section, cub. ft.	42
Equivalent in tons of each foot cut out	3·23
Average footage per month per drill on converted section	66.5

Notes

Total tons cut out in 7 months with average of 20 drills	31,087
" per month with 20 drills	4,441
" per drill per month during 7 months	221

During these seven months these twenty drills cost as under :—

	Cost per drill per month.
Spares	£9 12 4
Hose-mending	0 2 0
Fitters repairing; wages £30 per month	1 10 0
Engine-drivers running compressors; wages £22 : 10s. per month	2 5 7
Coal @ 18s. 4d. per ton	17 19 10
Lubricants @ 4s. 8d. per gallon	2 1 9

These figures do not give the full costs of running a rock-drill, but they are actual costs under the several heads mentioned. Mr. Seymour, in his estimate previously given, shows what the actual and complete cost is likely to be.

CHAPTER XVII

VENTILATION AND ILLUMINATION

Ventilation.—During sinking in vertical shafts ventilation is effected by bratticing off the pump- and ladder-way with thin planking, generally $\frac{7}{8}$-inch flooring boards; this compartment so formed is then used as the upcast. In most cases, with this provision, it is possible for men to descend again within half an hour after a blast. With the object of absorbing the bad fumes which may linger at the shaft-bottom, some water is often allowed to fall down the shaft to damp the broken rock lying in the bottom.

When the reef has been reached, and some development has been done, without, however, having effected communication with another shaft, a light galvanised iron pipe, generally about 22 inches in diameter, is often placed right away down the pumping compartment, and from it smaller pipes, about 10 inches in diameter, are led to positions as near the ends of the development drives as possible. In order to assist the air in ascending, this pipe is either taken as high as possible up on the headgear, or it is connected to an exhaust fan. In the deeper levels, where there is a great distance between any two shafts, this latter method will be used.

Where connection has been made between two or more shafts or openings to the surface, natural ventilation is relied upon, and, in the deep levels, efforts are made to assist it artificially. In all cases, the air from the exhaust of machine drills driven by air, serves to greatly simplify the

ventilation, for such air is introduced at the development ends, which are just the places that lie out of the line of natural ventilation. If the air were not introduced in this manner, it would probably have to be pumped down with a similar installation of mains and pipes, but under no or very little pressure.

In the outcrop mines the general ventilation is rarely planned. There are usually so many openings to the surface that there is no need for such precaution. In the City and Suburban the Main Incline shaft is generally downcast, but under certain conditions of the atmosphere it becomes an upcast. It is better in all cases for the main shaft to be downcast, because the bad upcast air materially contributes to rot the timbers. In the Robinson the West Main Incline shaft was the upcast, until connection was made through to the neighbouring mine, the Bonanza, when it became the downcast.

In the deep-level mines the ventilation will probably be relied upon to bring about a decrease in the underground temperature, so that, with the limited connection available from below to the surface, it will have to be more carefully planned.

On the Rand no local heat in the rocks themselves has yet been experienced. The slight amount of oxidation, which slowly proceeds when ore has been opened up, cannot appreciably affect the temperature of the air, and the dykes which cross the formation have not yet been found to bring any material amount of heat with them.

Some early determinations made by Mr. Hamilton Smith indicated an increase of 1° Fahrenheit for every 82 feet, but, owing to inaccuracies in the thermometers used, this result is in error. Mr. John Hays Hammond held that a smaller increase of temperature in depth might be expected, and the results of more recent determinations show that this is the case.

At a depth of 1877 feet, in No. 2 shaft of the Robinson Deep, the following temperatures were taken :—

Water in sump	73.5°	Fahrenheit
Drill-hole in South cross-cut . .	77.0°	"
" " West drive . . .	75.25°	"
" " West cross-cut in dyke .	77.25°	"

These temperatures were taken whilst the compressors were at rest, and whilst no men were working in the mine. The thermometers were left half an hour in the drill-holes.

In the No. 1 shaft of the same mine, no compressed air being in use, and the shaft forming at that time a dead end, so that the air was comparatively stagnant, the temperature at 2390 feet deep was found to be 78.5° Fahrenheit. Mr. M. Franke gave, in his annex to the State Mining Engineer's Report for 1896, the following interesting table of temperatures obtained in the deep-level shafts:—

Shaft.	Temperature about 50 Feet below Mouth of Shaft.	Temperature at Bottom of Shaft.	Vertical Depth of Bottom of Shaft.	Difference in Temperature.	Difference in Depth.	Depth for every 1° C. Rise in Temperature.
	Degrees Centigrade.	Degrees Centigrade.	Feet.	Degrees Centigrade.	Feet.	Feet.
Langlaagte Deep shaft No. 1 .	18.3	22.2	1342	3.9	1292	331
Langlaagte Deep shaft No. 2 .	17.7	21.1	1099	3.4	1050	308
Crown Deep shaft No. 1 .	17	21	1352	4	1302	325
Geldenhuis Deep shaft No. 1 .	21.1	22.2	941	1.1	892	810
Geldenhuis Deep shaft No. 2 .	16.7	18.9	928	2.2	879	400
Rose Deep shaft No. 1 . .	24.4	26.1	1201	1.7	1151	676
Rose Deep shaft No. 2 . .	24.4	26.1	1099	1.7	1050	616

N.B.—The Langlaagte shafts were not connected.
The Crown Deep shaft was connected with No. 2.
The Geldenhuis Deep shafts (as also those of Rose Deep) were connected.

From all these figures it is probable that the rate of increase in round figures is about 1° Fahrenheit for every 250 feet of depth.

The average temperature at 1000 feet appears to be about

72°, so that at 5000 feet, calculating upon a regular increase, it would be about 88° Fahrenheit.

As in all cases the temperature of the air will depend considerably upon that at the intake, the temperature experienced underground, more particularly in the main ways, should be, with efficient ventilation, appreciably less than that of the surrounding rock, except perhaps during some portions of the summer months.

It is generally considered that one man requires underground about 20 cubic feet of air per minute, but allowing for the air required to clear away the fumes of explosives, candles, etc., a much higher figure is usually allowed, the Prussian regulation, which allows 70 cubic feet of air per man per minute, being very frequently adopted, especially in collieries. In the deep levels it is probable that during one shift there will be as many as 1000 persons underground, so that these would, at this rate, require 70,000 cubic feet per minute. In these mines the shafts are large, having a section inside timbers of about 150 square feet, of which 100 square feet, more or less, will always be available for ventilation. Over this available area the average velocity of the air would be at the rate of 700 feet per minute, which is a very ordinary rate. This will probably have to be obtained by mechanical ventilators, and there are several fans on the Rand which are to be erected for various deep-level mines; these are all exhaust fans, the Walker fan being the one most in favour. It is well known that such fans are capable of dealing with very large quantities of air, 200,000 cubic feet per minute being by no means an unusual amount.

At present very little attempt is made to direct the air-currents underground. Sometimes a door is placed across the drive, and by leaving solid pillars or stulls through the old stopes near the surface, air-ways are kept intact. In the deep levels, systems of directing and regulating the air,

somewhat like those now in use in all large collieries, will probably have to be introduced.

The inclination of the reef and the regular system of development are favourable to good ventilation. The drives are advanced, and at regular intervals connection is made to the level above, by which means the circulating air is brought nearer to the advancing face. The distance between winzes in the deep levels is about 400 feet, and for this distance the ventilation caused by the exhaust from the air drill is sufficient. Where two drives on the same level have been advancing in opposite directions to meet, a greater distance than the above has often been driven without other ventilation than this exhaust.

In rising, it is generally considered that 150 feet from the level below is the limit beyond which it is not economical or warrantable to proceed because of the bad air.

Illumination.—In the larger mines the main stations, main cross-cuts, pumping stations, etc., are lit by small incandescent electric lamps, but in the mines proper, paraffin candles are universally employed, with the single exception of the City and Suburban, which mine possesses an installation of small oil lamps, although the advantage of using these is doubtful. It must be remembered that neither oil nor tallow is a suitable material for illuminating a gold mine, as even a small amount of such greasy material dropped among the ore would interfere most seriously with the process of amalgamation in the stamp mill, and might cause losses of gold out of all proportion to the amount of saving that could be effected by the use of a cheaper illuminant. Practically only two makes of candle are in use, those of the Price Candle Company and De Roubaix. The former are put up in 1 lb. packets of 8 or 10 to the lb., there being 25 packets in a case of about $\frac{3}{4}$ cubic foot capacity. The De Roubaix candles are put up loose, and not in packets, in boxes of about the same size; their candles are more opaque than the

former, and harder, never getting so soft as to bend. At the same time they are more brittle and appear not to give quite so good a light as the first-named.

A certain amount of care has to be exercised in giving out the candles to the boys from store, this being only done on the order of a foreman. In some mines the natives are compelled to show the pieces of candle that they bring back; larger pieces are taken from them and given out again next day, but they are allowed to keep small ends and to take them to their compound. The natives use candle-holders made of a piece of wire about $\frac{1}{8}$ inch in diameter and 3 to 4 feet long, one end of which is bent into a spiral so as to hold the candle, whilst the other end is pointed and bent into a hook (see Fig. 182), by means of which the candle can be hung up, any little crevice or projecting rock being taken advantage of for this purpose. In moving about the mine the boy holds the hook end of the wire, so that the candle is carried close to the ground, on to which it throws a good light. The white miner prefers a candle-holder such as is shown in Fig. 181, which is a very convenient form underground.

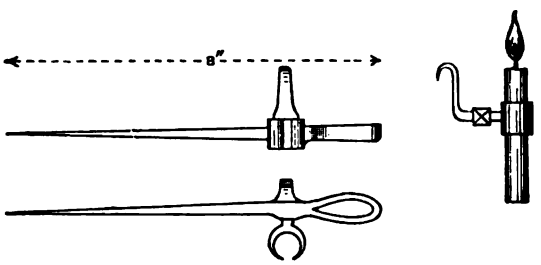


FIG. 181.—Miners' iron candle-holder.

FIG. 182.—Kaffirs' wire candle-holder.

In underground dynamite stores, incandescent electric lights are always used.

It is scarcely possible to calculate separately the proportion that the cost of illumination bears to the total cost of mining. From the Report of the Chamber of Mines for 1897 it seems that the consumption of candles on

the Rand was about $3\frac{1}{2}$ millions of pounds, whilst $5\frac{1}{3}$ millions of tons of ore were raised. These two figures do not apply to identically the same mines, and it is impossible to distinguish between sinking, development work, and mining, so that all that these figures can serve to show is that the total cost of illumination in all mining operations is about 4d. per ton of ore raised.

CHAPTER XVIII

TRANSPORT AND TRAMMING

THE transport of the ore, both underground and on the surface, is effected in four-wheeled trucks or waggons, which run on railed tracks.

It is usual to speak of transport on surface as "transport proper," and of underground transport as "tramming."

Rails.—The rails of the track are similar in section to railway metal, and, like these, are made of steel. The weight of rail used depends upon the weight per wheel running over them, upon the amount of traffic, and upon the speed. Ordinarily the speed is slow, and the following are the usual weights of rails used on the surface:—

Weight of full truck up to . .	1½ tons.	3 tons.
Weight of rail per yard . .	12 lbs.	14-16 lbs.

For incline tracks heavier rails are required.

Underground, the weight ordinarily used along the levels is 12 lbs. per yard, but in the main cross-cuts or other main tramming ways, heavier rails are laid.

On the surface, where small steam or electric locomotives are employed, a minimum weight of 22 lbs. per yard is required.

The sleepers are generally of rolled steel plate, and are trough-shaped. The chairs for the rails are often formed by a lug pressed out of the crown of the sleeper on one side, and by a loose clip and bolt on the other side, as shown in

Fig. 183, or by the arrangement indicated in Figs. 184 and 185.

The gauge of the track is usually 18 inches or 24 inches.

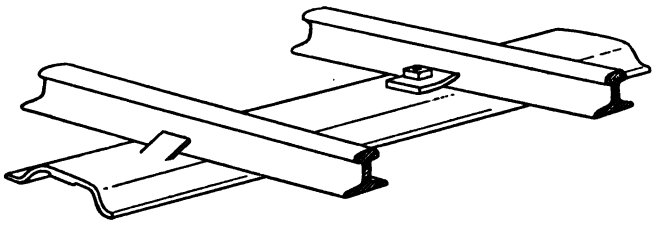


FIG. 183.—Rails and sleepers of an ordinary level track.

Trucks.—The double-side "V"-tip truck shown in Fig. 184 is the type which is more used than any other, though

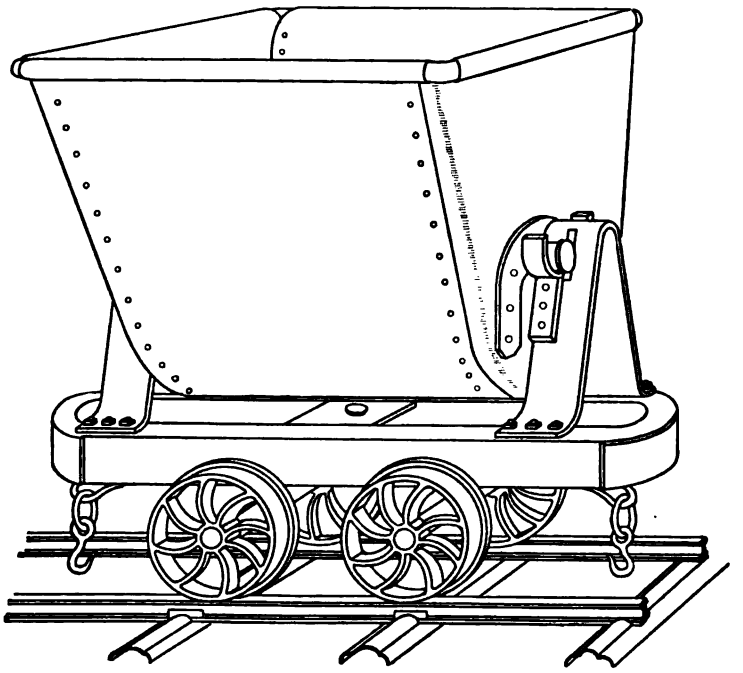


FIG. 184.—Double-side "V"-tip truck (Howard, Bedford, England).

underground the double-side "round"-tip truck shown in Fig. 185 is largely used, because it has a considerable capacity contained within the smallest over-all dimensions. It is likely, however, that in the deep-level mines where the drives are

larger, that the " V " tip will be used almost exclusively underground, because the shape allows the load to be more readily and completely discharged. The following description of these tubs is taken from the catalogue of Messrs. Howard, Farrar, & Co. :—

" The tip waggons are made with steel tubs carried upon two sets of wrought-iron trunnions. These trunnions are firmly riveted through the end plate of the tub, which is

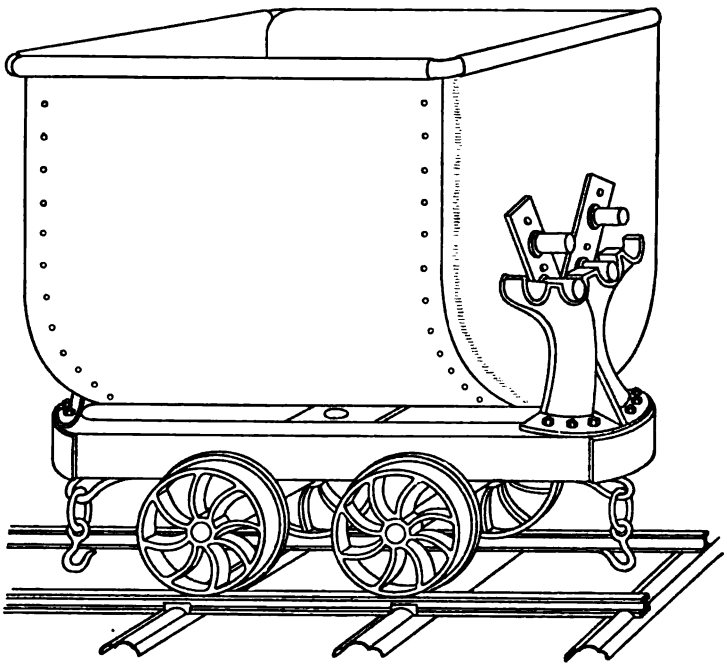


FIG. 185.—Double-side "round"-tip truck (Howard, Bedford, England).

strengthened by an additional inside and outside trunnion plate. The tub ends are flanged by hydraulic machinery, and are riveted firmly to the sides, which are of one plate. A strong, half-round, welded ring is riveted round the top of the tub and holds it rigidly together, or the end and side plates are bent over a round iron bar. By this method of construction a very strong and durable tub is obtained. The under-frame is of channel steel, which, in the latest pattern truck, is bent round to form bowed buffer ends, as shown in

Figs. 184 and 185. These ends are well stiffened, and steel angle stays are carried across the under-frame. The draw-gear is attached to a through stay. The axles are of steel, and the wheels of chilled iron or cast steel. Waggons with round buffer ends are far superior for light railway work to those built with corner buffers, as the liability to derailment on roughly-laid lines with sharp curves is greatly decreased."

The wheels are made with deep flanges and wide treads for running on roughly-laid lines. They are keyed on axles of mild steel, and these run in axle-boxes, which are generally made in the form shown in Fig. 186, which consists of a simple cast-iron fixed pedestal with a grease-chamber at the top, the axle being kept in place by a simple split cotter pin.

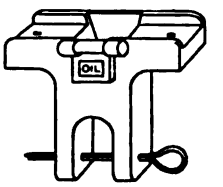


FIG. 186.—Ordinary form of axle-box.

The ordinary capacity of the truck, which is to be pushed along by hand labour, whether on surface or underground, is either 12 or 16 cubic feet. Of these two the smaller capacity is seen more frequently underground, because neither the facilities for manipulation nor the track are as good there as they are on the surface. Mechanical haulage is not at present much used underground, so that larger trucks than these are not often seen there; but on the surface, trucks with a capacity of 20 or more cubic feet, 1900 lbs. of dry ore or more, are used on the mechanical haulages.

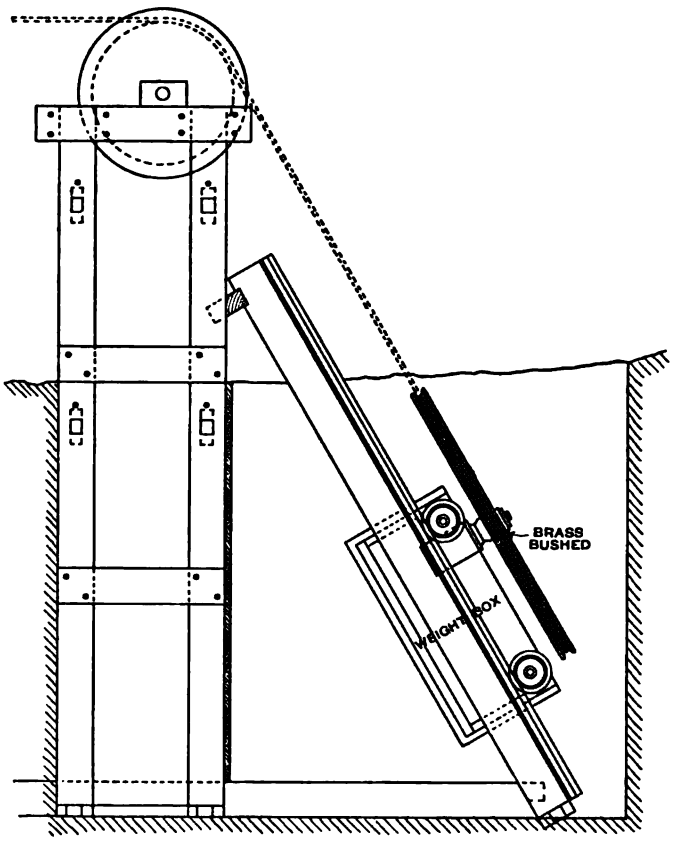
Where large trucks are employed with locomotives a central spring buffer and spring draw-gear are used.

In the Durban-Roodepoort mine, and in some of the adjacent mines, low wooden rectangular trucks are used, in which the wheels are rigidly fixed to the tub, so that tipping can only be effected by the use of such a tippler as is shown in Fig. 103.

In the Simmer and Jack, large railway trucks are used for the transport of ore on the surface, each of which is capable of holding several tons.

In addition to these side-tipping trucks, a few which tip

all round are in use for special work on the headgears, dumps or elsewhere, and others, from which the contents are discharged through hinged doors running along the lower portion of the sides, are occasionally used on the surface, when the truck is run to a position immediately over the bin.



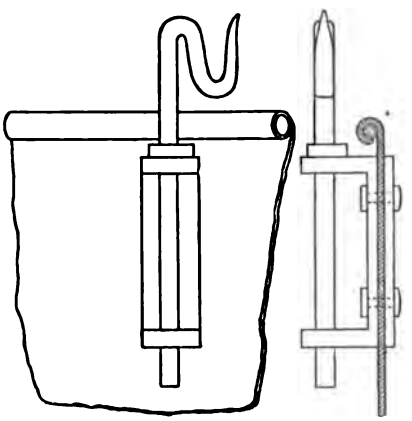
END ELEVATION.
Scale, $\frac{1}{4}$ inch = 1 foot.

FIG. 187.—Tension carriage and weight box.

As stated before, practically the whole transport of ore underground is done by hand labour. On the surface, however, the greater portion is done by one or other form of mechanical haulage. The most common form is the endless rope. Where it can be arranged to take the ore in a straight line along a track which is not too steeply inclined, this

method is considered the best; these conditions can in most cases be complied with.

The endless rope is carried along the hauling track so that the forward and return portions are from 5 feet to 6 feet apart, the distance required being such as will give sufficient clearance between passing trucks. It passes at one end round a fixed pulley, and at the other end round a pulley carried upon a tension carriage, which runs on its wheels along a track, and which has attached to it a suspended weight which keeps the rope sufficiently taut. Such a tension carriage, which forms also the weight box, is shown in Fig. 187, running upon a steeply inclined track. The movement of haulage is communicated to the rope generally through the fixed pulley at one end, though it may be done at any desired point. The rope is carried on pulleys only at proper stations. At all other points it either rests on the trucks or upon hollow cylindrical cast-iron rollers placed at regular distances, from 75 to 100 feet apart.



Scale, $\frac{1}{10}$

FIG. 188.—Jockey fixed on to the end of a truck for mechanical wire-rope haulage.

The rope itself is of steel wire. Its size depends upon the amount of strain to which it is to be subjected, and this depends again upon the inclination of the track and the amount of ore to be hauled. The usual size for 16 cubic feet trucks, running on a track which for the greater part is horizontal, is $\frac{5}{8}$ inch diameter. For 20 cubic feet trucks, and on an incline, as at the Geldenhuis Estate, larger ropes are required.

The trucks are made to grip the rope by a jockey, which is a fork so fixed on one end of the truck that it can revolve about a vertical axis, as shown in Fig. 188. This axis is not

quite in line with the rope, nor is the fork, which, when extended in that direction, is about as much on one side of the rope as the axis is on the other. At the loading station the truck is run in under the rope and the rope is pressed down so as to pass through the fork. In its endeavour to regain its proper line, it turns the fork about the vertical axis, and by being slightly bent is securely held in the fork, the grip being stronger the greater the tension of the rope. At the off-loading or "knock-out" station the rope is generally lifted out of the fork without difficulty as it rises to pass over a pulley, but at this station it is usual to have a native to knock the rope up. The usual rope speed is from 150 to 250 feet per minute, and the trucks are, as far as possible, placed at regular distances apart, generally about 50 feet. This method is in use at a good many mines. At the Geldenhuis Estate the length hauled is about 4500 feet up a slight incline, which makes the total height attained about 115 feet. The gauge of the track is 22 inches and the weight of the rails is 20 lbs. per yard. The trucks are of 20 cubic feet capacity, and they are placed at about 50 feet apart. This plant is run at such a speed that all the ore milled day and night —about 600 tons—is easily hauled during the day shift of ten hours.

This method is not used where the truck passes from the horizontal to an incline which is greater than 1 in 6, for at the junction with the gradient the rope, being unable to follow the angle, hangs in a curve which is far enough above the track to pull the rope out of the jockey. This, however, might be remedied by making the track follow the curve. For inclinations greater than 1 in 6 endless chains working very similarly are used. The fork of the jockey in this case is rigidly fixed on to one end of the truck and the chain is slipped into it so that one link is caught. By this method hauling is being effectively done on inclines of 1 in $2\frac{3}{4}$.

Wherever possible, wire rope is used instead of chain,

because of the great wear and tear with the latter, because the weight is so much greater, and because it is not possible to keep chains nearly as taut as ropes, as the links have no rigidity between themselves, and in consequence they are for the greater part dragging along on the track.

In the Durban-Roodepoort an endless rope mechanical haulage is used to transport the tailings, in which the rope passes underneath the truck body, but this method of under-tub haulage is unusual.

Where the track is curved or where public roads cross the line of transport so that, if mechanical rope haulage were used, it would be necessary to construct expensive bridges, the ore is transported in trains of trucks which are either pulled by engines or by animal power.

Animal power is now being rapidly superseded by electric engines such as are in use at the Crown Reef, New Modderfontein, Violet, and Wemmer mines. These engines are of the type in which the power is communicated by an overhead wire. They run the train at much higher speeds than are used in mechanical haulage, the ordinary speed being about 8 miles per hour. At the Wemmer mine the haulage with animal power (mules) has just been superseded by electric power. With mules there were five trains, of eight trucks each, always in use, and there were also two spare trains. The distance trammed was about 6000 feet, in which there was a drop of 84 feet, so that the loaded trucks, after a certain distance, were run down with the brakes. To bring the trucks back two mules were required for each train. The trucks were of 20 cubic feet capacity, the gauge was 24 inches, and the weight of the rails was 16 lbs. per yard. In this long stretch there was only one crossing.

During the half-year ending 31st August 1896 the cost of transport per ton was 10.098d., and during the previous half-year 7.230d., the difference being chiefly due to the increase of the price of fodder for the mules. With the electric

engines the trains will be longer, the speed of running greater, and the capacity of the trucks will be larger—30 cubic feet,—so that there will not be so many trucks in use. It has been estimated that the cost of hauling will not exceed half of the previous cost.

At the Crown Reef the ore is transported in trucks drawn by an electric engine, a distance of about 4500 feet from the main shaft to the mill, in which distance there is a drop of about 70 feet. The transport on this track cost during the year ending 31st March 1897 at the rate of:—

Working costs per ton	1.461d.
Maintenance „	1.947d.
	3.408d.

During the same period the underground tramming, which is done by natives, cost at the rate of 1s. 5.854d. per ton, and the greatest distance along which any ore was so trammed was not above 1500 feet, the average distance being much less; this cost is perhaps high, because it is generally found that tramming and loading in the drives may be contracted for at from 5d. to 6d. per load of 1800 lbs., though this figure covers but little more than the labour, and is far from being the inclusive cost.

At the Geldenhuis Estate during the six months ending 31st March 1893, when the transport was done by mule haulage, it cost 1s. 0.91d. per ton; whereas during the year ending 31st March 1896, with the mechanical haulage equipment which has been described, it was 5.35d., and during the following year the sorting and transport together cost at the rate of 10.13d. per ton.

At the Simmer and Jack the ore will be transported from the main shafts to the mill by steam traction in trucks running on rails of large gauge; from the first row of deep-level shafts the up-gradient to the mill will be one of 2½ per cent.

At the Lancaster the ore will be similarly transported. The following is the equipment at this mine:—

Details of Locomotive—

Diameter of cylinders	$5\frac{1}{2}$ inches
Stroke	$10\frac{1}{4}$ "
Diameter of wheels	$22\frac{3}{4}$ "
Gauge of track	24 "
Approximate weight when loaded with coal and water	11,700 lbs.
Approximate weight when empty	9,900 "
Gross load conveyed on a level	80 tons
" " " gradient 1 in 500	70 "
" " " " 1 in 200	50 "
" " " " 1 in 100	30 "
Speed on a level track	$6\frac{1}{4}$ miles per hour

To run with this engine, there are "V"-tip trucks of 20 cubic feet capacity, 15-inch wheels, and bodies of $\frac{1}{8}$-inch steel plate.

It often occurs that the ore arrives at the mill site on the surface level, so that it has to be raised to the bins; this is generally done by an elevator, that which is known as Reedy's elevator being most frequently used.

CHAPTER XIX

ORE-DRESSING

BEFORE sorting was adopted, the rock hauled was divided by screening into "fines," which went direct to the mill, and into coarse ore, which was first crushed and then milled, the crushing being considered a part of the milling operation. With the extensive use of sorting, these preparatory operations of screening, sorting, and crushing are best included under the term "ore-dressing," for with the adoption in some cases of tandem crushing and double sorting, crushing will be undertaken in part with the object of obtaining improved sorting, and not entirely to assist milling.

The three ore-dressing operations mentioned above are, wherever possible, done one above the other, so that the ore falls from one to another, as illustrated in Fig. 189, where they are all shown as being conducted on the headgear.

It is perhaps the better practice, in order that the operations of sorting and crushing may not be cramped for want of space or height, to do the first screening alone on the headgear, and to conduct the other two operations at what is called a "crusher station."

This may be attached to the headgear, but resting on its own foundations, or where there are two or more headgears it may be centrally arranged between them. It is more or less immaterial at what level it is placed, as long as care is taken that no ore is needlessly raised, but sinking it into the ground so that its upper floor is reached by a gentle incline

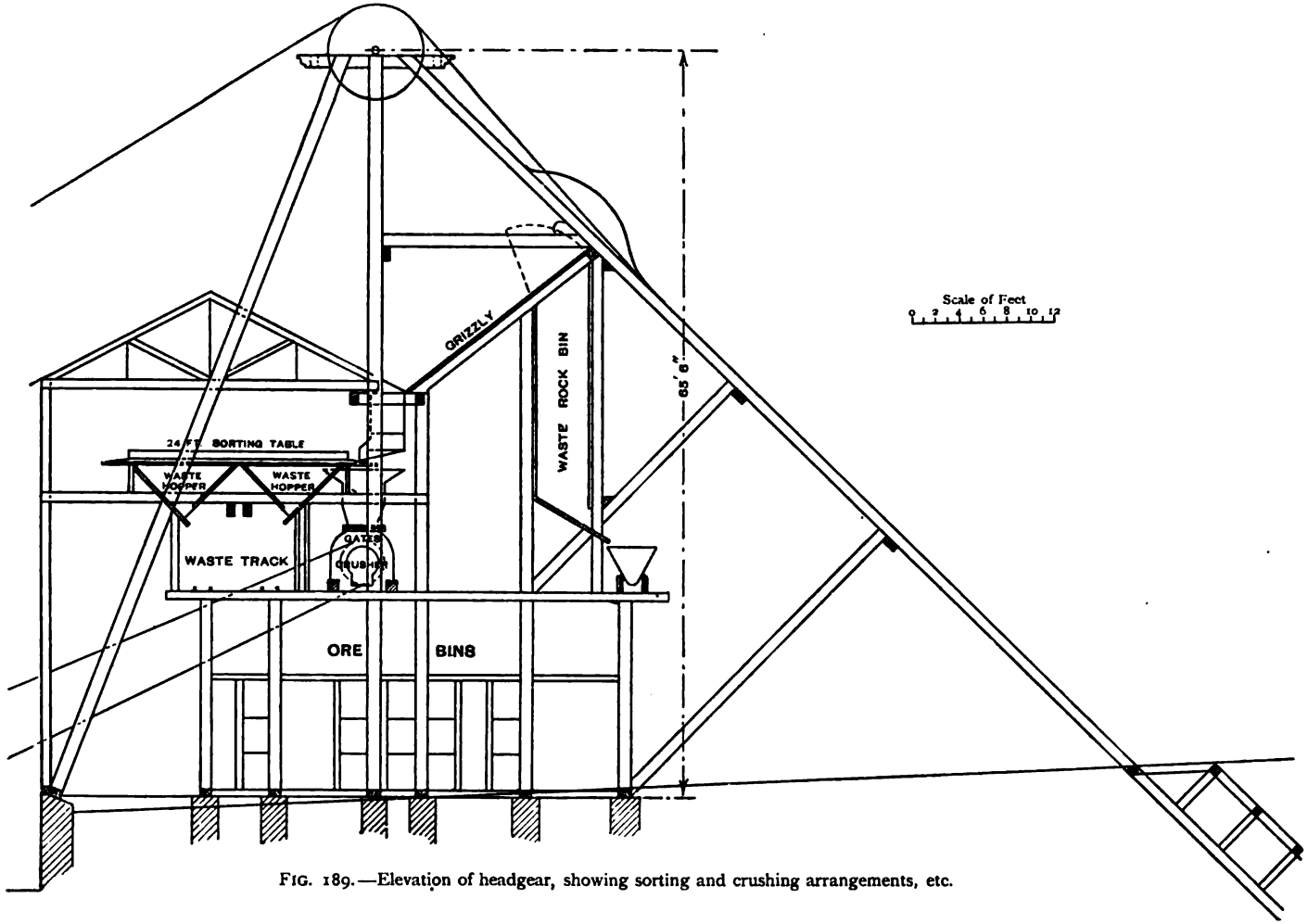
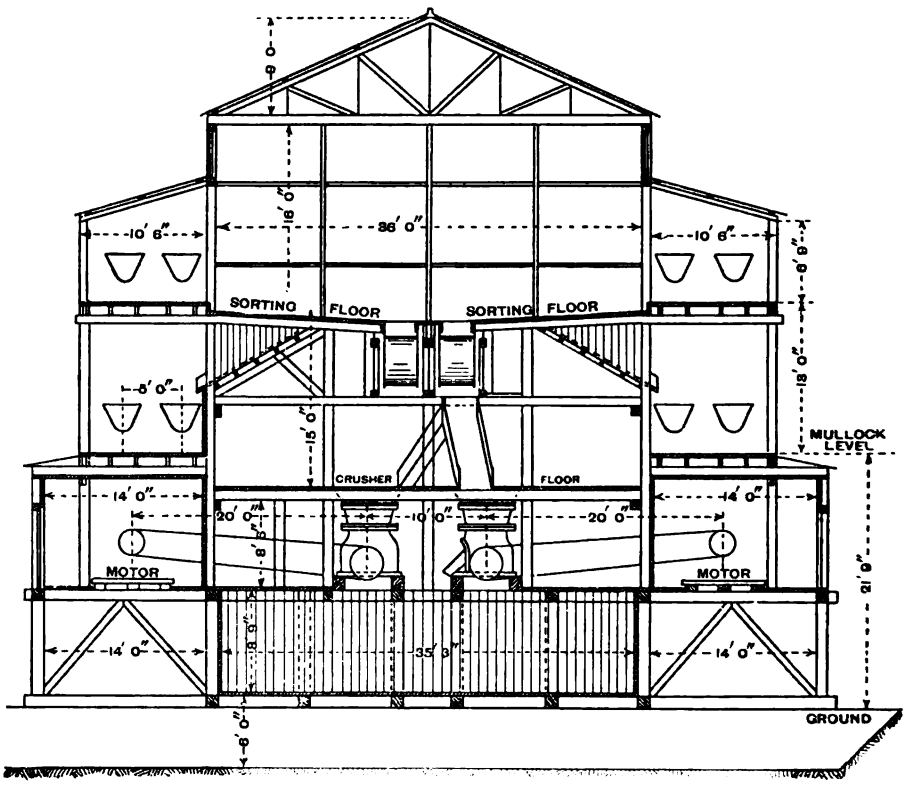


FIG. 189.—Elevation of headgear, showing sorting and crushing arrangements, etc.

from the headgears along which mechanical haulage may work, affords greater stability to the structure.

The advantage of having bin capacity between the first screening and the sorting is that the latter can be done regularly and need not depend upon an irregular supply drawn directly from the mine. Thus in the Geldenhuis Estate the



Scale, $\frac{1}{16}$ inch = 1 foot.

FIG. 190.—Elevation of crusher station.

whole of the ore which has been drawn through the night, together with that drawn during the day, is sorted in the daytime, resulting in an appreciable decrease in cost and in improved sorting.

A complete crusher station arranged with sorting floors, carrier belts, and Gates crushers is shown in Fig. 190. In this arrangement the coarse ore to be sorted is brought in trucks

from the headgear by mechanical haulage on to the top platform of the station, from which it is tipped on to slightly inclined sorting floors, where it is cleaned with water, the waste picked out and removed to small bins, and the ore is thrown up on carrier belts, which take it to the crushers. Beneath these crushers there are capacious bins under which trucks run to be filled with ore for the mill. These carrier belts are not held in any favour; it is better, where sorting floors are used, to take care that the Gates crushers are conveniently arranged so that the sorted ore may be shovelled into them; in other respects the figure represents very well the sequence of ore-dressing operations at a central crusher station.

Screening.—A description of screening is given under sorting.

With the adoption of tandem crushing and sorting, the first screening will be effected on the headgear, the second after the first crushing, and the third and final after the second crushing. The same-sized grizzly will be used in each case, and at each operation a proportion of fines will be separated.

Sorting.—The process of sorting consists of separating, as far as possible, the waste rock from the reef matter with which it has been broken. This waste matter is generally quartzite, though in the Van Ryn and Nigel districts a good deal of slate occurs in addition. It is easy to distinguish by appearance such waste from the banket, especially when both have been cleaned. No attempt is made to pick the good from the poor banket, because the most experienced samplers can only at best make a good guess at what the grade of any piece of reef might be by its appearance.

The proportion of waste broken with the reef depends upon the thickness of the reef, upon its nature, and also upon the bedding planes which occur near the reef.

Where the reef is small it is necessary that sufficient waste

be taken out to provide a stope of a size which shall be economical for mining.

Where a reef consists of leaders separated by quartzite, it is necessary to mine the quartzite so as to carry all the leaders, and where two reefs are so close together that the piece of quartzite between them will not permit of two stopes being made, it is also necessary to mine waste.

Sometimes above the reef there are "false backs" of waste, which, when they are small, are taken down for safety, though when they are large they are kept up by pillars. Similarly, the rock often breaks from bedding planes in the floor, and quartzite is brought away with the reef.

It may be said that waste is of necessity being mined all along the Witwatersrand, the maximum amount being got with the rich and small reefs, as the South Reef on Roodepoort, and with those which are split up into several leaders, as the South Reef at Johannesburg, and the minimum amount with the large and compact reefs, such as the Main Reef of the Langlaagte Estate. With the more extended use of rock-drills in the stopes the proportion of waste mined will be increased. In some cases it amounts to as much as 80 per cent of the rock broken, and it is probable that the minimum amount is not less than 10 per cent.

Gold does occur to some extent in the quartzite, but generally the amount is too small to be material. With the Chimes Reef the gold in the quartzite is more considerable. In the Main Reef at Johannesburg there is often as much in the quartzite partings as there is in the banket, and the quartzite which comes between two rich leaders of the South Reef does also in places contain gold. In these and similar cases, before sorting is determined on, the value of the quartzite must be found out, and the best way of doing this is to sort out the quartzite and carefully sample it. The amount of quartzite broken with the reef which does contain a material

amount of gold is, however, so small that, after this mention of the occurrence, it will be neglected.

From the start it has been the rule to separate the larger pieces of waste underground and to pack them in the stopes to form stulls, but on account of insufficient light and the covering of dirt on the ore, it is not possible to carry such sorting to a satisfactory extent, and, in consequence, all the ore which is brought to the surface contains a considerable amount of waste. As this has been mined and hauled, no further expense is incurred on that account, but the complete reduction of the ore, in order to recover the ordinary percentage of extraction obtained along the Rand, costs, for milling and cyaniding, etc., at present about 8s. per ton. As the value of the gold which would be recovered from clean waste is a negligible amount, it may be considered that each ton of such rock which passes through the reduction processes incurs an expenditure of about 8s., for which there is no return. Hence the necessity for sorting.

It is now realised that there is a direct saving of the gold effected by sorting, in addition to the saving of the reduction expenses, because the waste which has been sorted out does not often assay as much as 1¼ dwt. of fine gold per ton, whereas the residues from the different reduction processes generally assay more. From this fact it follows that if the waste, instead of being sorted out, were crushed and treated, it would arrive at the residue dumps with a higher assay than it originally had (see Appendix I., p. 478).

Another advantage of sorting is that the capacity of the mill and the reduction plants to treat auriferous banket is increased proportionally to the amount of waste sorted out.

The accompanying statement, p. 410, showing the advantage of sorting as experienced at the Ferreira, is taken from the General Manager's Report, 31st December 1896.

[TABLE

Ferreira Gold Mining Company, Limited

Statement Showing Advantage of Sorting

Period.	Ore mined.	Waste Rock picked out.	Ore milled.	Percentage of Waste Rock picked out.	Value of Ore mined.	Value of Ore milled.	Increased Value of Ore milled due to Sorting.	Value of Additional Gold won due to Sorting.			Direct Saving due to Sorting.		
	Tons.	Tons.			Dwts.	Dwts.	Dwts.	£	s.	d.	£	s.	d.
Year ending March 1893 . . .	57,794.5	9,017.5	48,777	15.6	19.38	22.69	3.31	28,899	17	4.80	6,825	4	1.84
Half-year ending September 1893 .	35,953	11,217	24,736	31.19	19.70	27.95	8.25	36,526	14	9.60	9,360	3	9.65
Half-year ending March 1894 .	34,259	11,782	22,477	34.36	22.491	33.495	11.004	52,599	17	9.36	10,282	0	9.62
Nine months ending December 1894 .	66,543	29,759	36,784	44.72	19.33	33.762	14.432	109,547	4	5.39	14,656	13	4
Year ending December 1895 . .	99,689	38,435	61,254	38.50	19.423	30.978	11.555	150,405	7	4.47	22,966	9	4
Year ending December 1896 . .	178,918	58,156	120,762	32.5	17.256	25.047	7.791	199,932	1	1.83	27,843	9	6
Totals . .	473,156.5	158,366.5	314,790					577,911	2	11.45	91,934	0	11.11

The assay value of the waste rock picked out during the year averaged 1 dwt. 0.19 grain gold per ton.

In addition, the following tabulated statement shows the rise in the value of the ore, indicated by the value of the gold recovered from the plates, which followed in the Geldenhuis Deep and Crown Reef mines on the adoption of sorting :—

	Geldenhuis Deep.		Crown Reef.	
1897.	Value of Gold recovered on Plates per Ton of Ore.	Percentage sorted on Surface.	Value of Gold recovered on Plates per Ton of Ore.	Percentage sorted on Surface.
	s. d.		s. d.	
January .	21 1	Nil.	28 2	Nil
February .	20 10	,,	32 4	,,
March . .	21 6	,,	30 4	,,
April . .	24 1	12-15 per cent	38 5	15.2 per cent
May . .	23 7		33 11	13.6 ,,
June . .	27 4		33 5	16.7 ,,

As long as the assay value of the waste sorted out is less than that of the residues from the reduction processes, and the cost incurred in sorting a ton of waste rock is less than the cost which would be incurred in reducing it, the process of sorting is not being carried too far.

In practice it is found that the waste sorted out is not absolutely clean, there being always some small admixture of banket, the gold of which makes, with that of the quartzite, an amount which can be appreciated by assay, so that the waste has to be regularly sampled in order to ensure against any undue loss.

The slimes which result from reduction have any assay value which is proportionate to the value of the ore treated, and this being the case, the value of any ore might be so increased by sorting, that the resultant slimes might be treated at a profit, whereas, had there been no sorting, they might be of too low a value. Another advantage is that during the sorting process a good many hammer and pick heads, pieces of iron and wood, etc., are picked out, which otherwise would

pass through the crushers and into the mortar-boxes, increasing the chances of breakage of the milling machinery. The items under which extra cost is incurred by sorting, and those under which direct saving is effected, may be thus tabulated :—

Extra Cost.	Direct Saving.
Cost of sorting.	Cost of transport.
Depreciation on sorting plant.	,, ,, crushing.
	,, ,, milling.
	,, ,, cyaniding, etc.
	Depreciation on plants.

The cost of sorting increases with the greater extent to which it is carried; for instance, at the Ferreira when, with grizzlies set with 2¼-inch spaces, 25 per cent of the rock hoisted was sorted, the cost was only about 6d. per ton sorted out, but latterly, with 1-inch spaces, and sorting about 35 per cent, the cost has been about 1s. 6d., and Mr. Johns, the General Manager, has stated as the result of a test, that if the distance between the grizzly bars were lessened to ¾ inch, a further profit of £250 per month could be made after deducting 2s. 6d. per ton as cost of sorting the extra quantity, and £25 per month as depreciation of the extra sorting plant required.

The cost of sorting at the Crown Reef during the months of April, May, and June 1897, just after the plant was started, and when extraordinary maintenance was incurred, was only 3s. 9.683d., and the total cost of reduction during the same period was approximately 7s., so that the amount saved per ton sorted out was at least 3s.

Underground Sorting.—Where a considerable amount of waste is broken, the larger pieces can be effectively sorted underground, either to be packed in the working stope to form stulls, or to be trammed to other stopes which have been worked out. This sorting will be a greater consideration in deep levels, where it will not be so easy to run down waste

from the surface for purposes of filling, and it is described under stoping.

Following the accepted range of the term, sorting proper is taken to include surface sorting only.

Principles of Sorting.—The practice followed is to separate, by tipping the ore on to inclined grizzlies, that portion of the ore to be sorted from that called "fines," which is regarded as being economically unsortable.

This is generally done in one stage, or more rarely in two stages. When in two stages, the ore is first tipped over grizzlies set, say with 3-inch spaces, and those pieces which do not pass through are subjected to a sorting which for them is final, whilst those pieces which pass through fall on to a second grizzly set with smaller spaces, and again that portion which does not pass through is sorted, and that which does constitutes the "fines." Provision is being made in some of the deep-level mines to vary this principle by first crushing the ore coarsely and subjecting the pieces which do not pass through a grizzly to a sorting which is not final, but which is completed after a second and finer crushing has been made. As, however, for the waste thrown out, each sorting is final, care has to be taken that only clean waste is rejected; the second crushing breaks up the ore so that a further amount of clean waste is detached. With this practice, which is known as "tandem crushing," and which would appear to be the more commendable, only one size of grizzly will be used and some fines will be separated at each crushing.

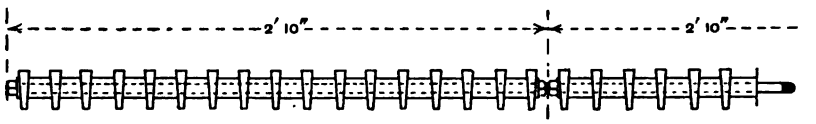
Methods of Sorting.—The sorting is effected:—

1. On a sorting floor.
2. On a revolving table.
3. On a moving belt.
4. On a shaking table or shoot.

In connection with all of these the grizzly is used. It consists of steel bars placed parallel and at regular distances apart. In order to perform its work effectively, it should be

set at an angle of not less than 40°, an angle of 45° being about the best. It should be a good length, viz. from 13 feet to 15 feet, and properly supported and braced. In order to keep its surface as free as possible, the bars in section should be tapered downwards, so as to minimise the chances of pieces of rock becoming lodged between them.

The size of the bars depends upon the distance they are placed apart. The following dimensions are those of a grizzly set with $1\frac{1}{4}$-inch spaces: Bars 14 feet 6 inches long, $2\frac{1}{2}$ inches deep, $\frac{3}{4}$ inch wide on top, and $\frac{1}{2}$ inch at bottom. Seventeen of these bars are bound together to make a section 2 feet 10 inches wide; they are braced by passing long bolts through them, two of which are placed a foot each from either end, and two others at regular distances between them;



Scale, 1 inch = 1 foot.

FIG. 191.—Section of grizzly.

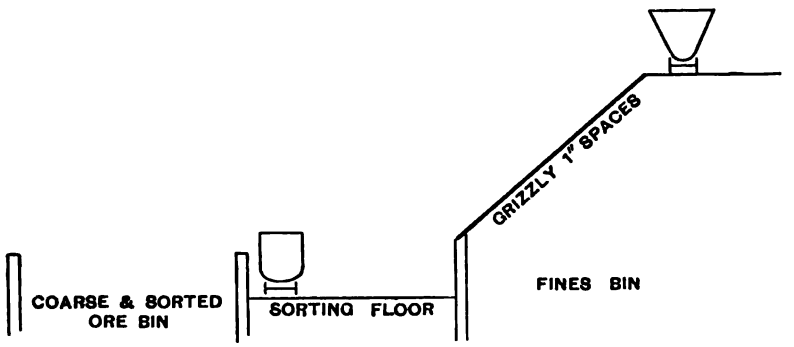
along these bolts the bars are placed alternately with cast-iron distance washers, which keep them at the correct distance apart. A section across such a grizzly is shown in Fig. 191.

Where the spaces are larger, heavier iron bars, and in some cases rails are used, and where a greater extent of surface is required, two or more grizzlies are placed side by side, so as to gain greater width.

Sorting on Floors.—The following diagram (Fig. 192) shows roughly, in section, the arrangement of the sorting floor of the Ferreira.

The ore from the skip is dumped into a small bin, from which it is discharged into side-tipping trucks, which run along a floor at the top level of the grizzlies; from these trucks the ore is shot down over the grizzlies at any place wherever it is desired. These grizzlies are 13 feet long and are placed side

by side for an extent of 70 feet; they are set at an angle of 40°, and the spaces between the bars are an inch wide; the fines which pass through, amounting to about 30 per cent of the rock mined, are not sorted. At the bottom end there is a drop of about 3 feet to the sorting floor proper, which is 70 feet long and about 11 feet wide. There is a track running along this floor on which trucks run to receive and take away the waste picked out; the remainder of the floor is covered with ½-inch steel plates. All the ore which does not pass into the "fines" bin collects in a heap at the bottom of the grizzlies, from which it is spread out by means of a heavy two-



Scale, 1 inch = 10 feet.

FIG. 192.—Diagrammatic section across the Ferreira sorting floor.

pronged rake and cleaned by playing water on to it. The water main is placed about 5½ feet above the floor, along the far side; at equal distances along its length twelve nozzles, 30 inches long, are arranged with flexible joints, and the pressure of the water is about 18 lbs. per square inch.

When clean, the waste is separated and thrown into the waste trucks, which run it out on to a dump, whilst the sorted ore is shovelled into a bin about 12 feet wide, which runs along the other side of the sorting floor. Large pieces containing ore and reef are broken up with large hammers for closer sorting.

Generally about nine boys on each shift are actually engaged in sorting, and two work on the top, distributing the ore; there is a white man in charge of these.

The water from the floor runs into a tank at the foot of the headgear, and afterwards into settling-pits, where all the slimes settle; these are cleaned out periodically and the contents sent to the mill.

At this mine sorting was first systematically undertaken in May 1893. For the four months from June to September 1893, 26.53 per cent of the total rock hauled was sorted, at an average cost of 6d. per ton sorted out. The grizzlies were then set with 2¼-inch spaces.

When closer sorting was decided upon, these spaces were reduced to 1 inch, and with this width the following figures and costs have been obtained:—

Month.	Tons milled.	Tons sorted.	Total Tons.	Per cent sorted.	Total Cost of Sorting.			Cost per Ton sorted.	Cost per Ton milled.
					£	s.	d.	d.	d.
July 1896	10,709	6657	17,366	38.33	443	11	10	16	10
Aug. „	9,455	6256	15,711	39.81	443	7	3	17	11.2
Sept. „	10,565	4898	15,463	31.67	410	4	6	20	9.3
	30,729	17,811	48,540	36.69	1297	3	7	17.4	10.1

The total cost of sorting included all white labour, native labour, native keep, and stores, distributed as under:—

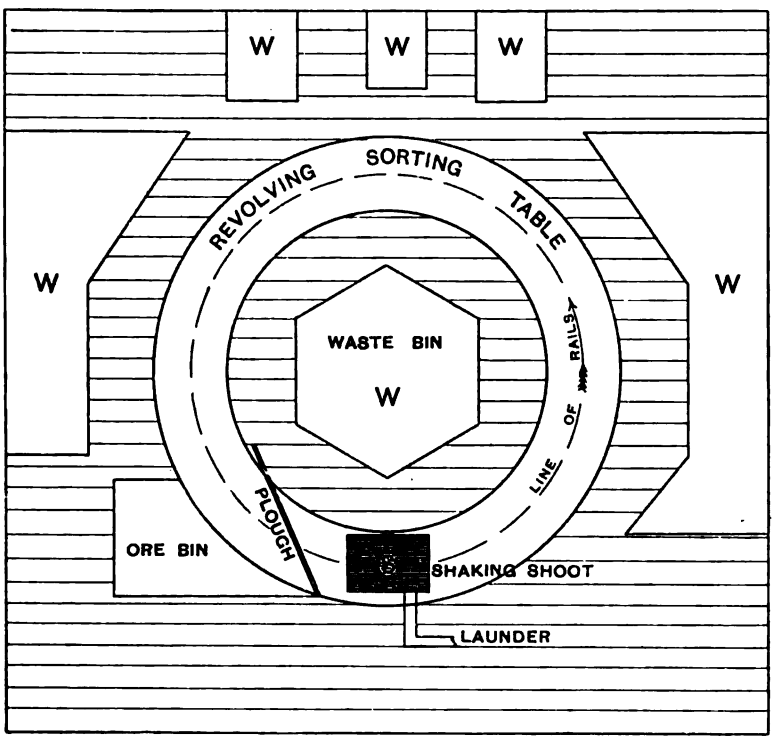
	July.			August.			September.			Total.		
	£	s.	d.	£	s.	d.	£	s.	d.	£	s.	d.
White labour .	61	10	0	56	10	0	60	0	0	178	0	0
Native labour .	287	2	9	291	16	9	266	11	0	845	10	6
Native keep .	72	13	5	72	19	2	63	6	1	208	18	8
Stores . .	22	5	8	22	1	4	20	7	5	64	14	5
	443	11	10	443	7	3	410	4	6	1297	3	7

The cost of the stores shown above is higher than it should be; £15 per month is about a correct expenditure for this item.

These figures show plainly the extra cost of closer sorting.

Sorting on Revolving Tables.—The following figure (193) is a diagrammatic plan of the circular table used at the Crown Reef.

The ore to be sorted passes from the small bin above, over a shaking shoot, S, on to the table, at a rate determined



Scale, 1 inch = 10 feet.

FIG. 193.—Diagrammatic plan of the Crown Reef revolving sorting table.

by a regulator at the mouth of the bin. The table has an outside diameter of 25 feet, and an inside diameter of 17 feet, so that it is 4 feet wide; its surface is covered with a top wearing plate $\frac{5}{8}$ inch thick, under which there is another plate $\frac{1}{4}$ inch thick; the surface inclines slightly downwards towards the outside edge, around which a launder runs. The whole table is supported on wheels 5 feet apart, which run on a

circular track underneath. On these wheels the table revolves about once in every 85 seconds, the motion being imparted near the periphery by means of a rack and pinion. Whilst the ore is on the shaking shoot it is cleaned by water, which is thrown sharply on to it from three rows of jets. The bulk of this water runs off at once into a launder, which takes it to a settling-tank beneath; the remainder, which goes round with the table, falls into the circular launder and is led back again to be discharged with the first portion.

Boys standing inside and outside the table remove the pieces of waste, and the reef is turned off into a bin, after an almost complete circuit has been made by a plough fixed as shown in the diagram; the large pieces which contain reef and waste are taken off by hand and broken up.

There are generally the following boys working per shift:—

2 boys sorting, standing outside the table,
2 „ „ „ inside „
1 boy at the regulator,
1 „ taking off the large pieces,
2 boys breaking up large pieces,

and there is one white man, who in addition looks after the crushing and the headgear generally.

This sorting table has fully come up to expectations as regards efficiency, but as it was added to an already existing headgear, which necessitates tramming the ore to the table, and raising the ore, after it has been sorted, to the rock-crushers, the working expenses are not as low as in some other plants.

The following table, kindly prepared for me by the General Manager, Mr. S. J. Jennings, shows the sorting statistics at this mine for the months of April, May, and June 1897. As the sorting table was only started just before this period, the maintenance costs are not normal, as they include the cost of various alterations.

Crown Reef Gold Mining Company, Limited

Sorting Table Statistics

Date.	Tonnage.			Assay Value.	Expenses.			Working Cost.		Maintenance Cost.		Total Cost.		Per cent of Rock mined.
1897-98.	Tons Waste sorted out.	Tons Ore sent to Mill.	Tons Rock hauled from Mine.	Per Ton Waste dwts.	Working Cost.	Maintenance Cost.	Total Costs.	Cost per ton on Waste Tonnage.	Cost per ton on Milled Tonnage.	Cost per ton Waste Tonnage.	Cost per ton Milled Tonnage.	Cost per ton Waste Tonnage.	Cost per ton Milled Tonnage.	Sorted out.
					£ s. d.	£ s. d.	£ s. d.	s. d.	d.	s. d.	d.	s. d.	d.	
April 1897	2309	12,686	15,104	1.333	254 10 10	34 8 11	288 19 9	2 2.457	4.815	0 3.580	0.651	2 6.037	5.467	15.2
May ,,	2436	15,669	17,953	0.958	252 10 3	337 4 1	589 14 4	2 0.878	3.867	2 9.222	5.164	4 10.100	9.031	13.6
June ,,	3014	15,682	17,951	1.625	290 0 10	308 3 7	598 4 5	1 11.095	4.438	2 0.539	4.716	3 11.635	9.155	16.7
	7759	44,037	51,008		797 1 11	679 16 7	1476 18 6	2 0.655	4.344	1 9.028	3.705	3 9.683	8.049	15.21

On the Treasury Incline Headgear the ore from the skips is dumped on to a grizzly, the bars of which are of railway metal set at an angle of 30° and about 6 inches apart. That portion which does not pass through falls down on to a floor, about 22 feet by 18 feet, covered with iron plates, where it is broken up with large hammers; the waste is picked out and removed to a bin, and the ore is passed down to be shovelled into a crusher. That portion which passes through falls immediately upon a second grizzly, set at an angle of 45°, and the spaces of which are 1¼ inches wide. The ore which does not pass through this second grizzly falls into a shaking shoot running along the bottom, which leads to a circular sorting table, and that which passes through constitutes the "fines."

This sorting table differs from that in use at the Crown Reef in that there is an upper shelf on the same structure on which the waste is placed, and in that it is driven by means of a central shaft; the sorting also is done by boys standing on the outside only.

The outside diameter is 24 feet, and one revolution is completed in 1½ minutes. At the bottom of the shaking shoot which leads the ore on to the table there is a row of water jets which clean the ore. As the table revolves the sorting is done, and when the greater part of the circuit has been made, the clean ore is forced off near to the crusher by a plough. The waste from the upper shelf is taken off by a shovel at a point a little farther on, as shown in the diagram.

The crushers and the sorting table are on their own foundations and are not supported by the posts of the headgear.

The arrangement of the shelves of the table is identical to that illustrated in Fig. 194, which has been designed by Messrs. Leggett and Wilson, of the engineering department of Messrs. Neumann and Company.

Sorting on Moving Belts.—Like all other belts in

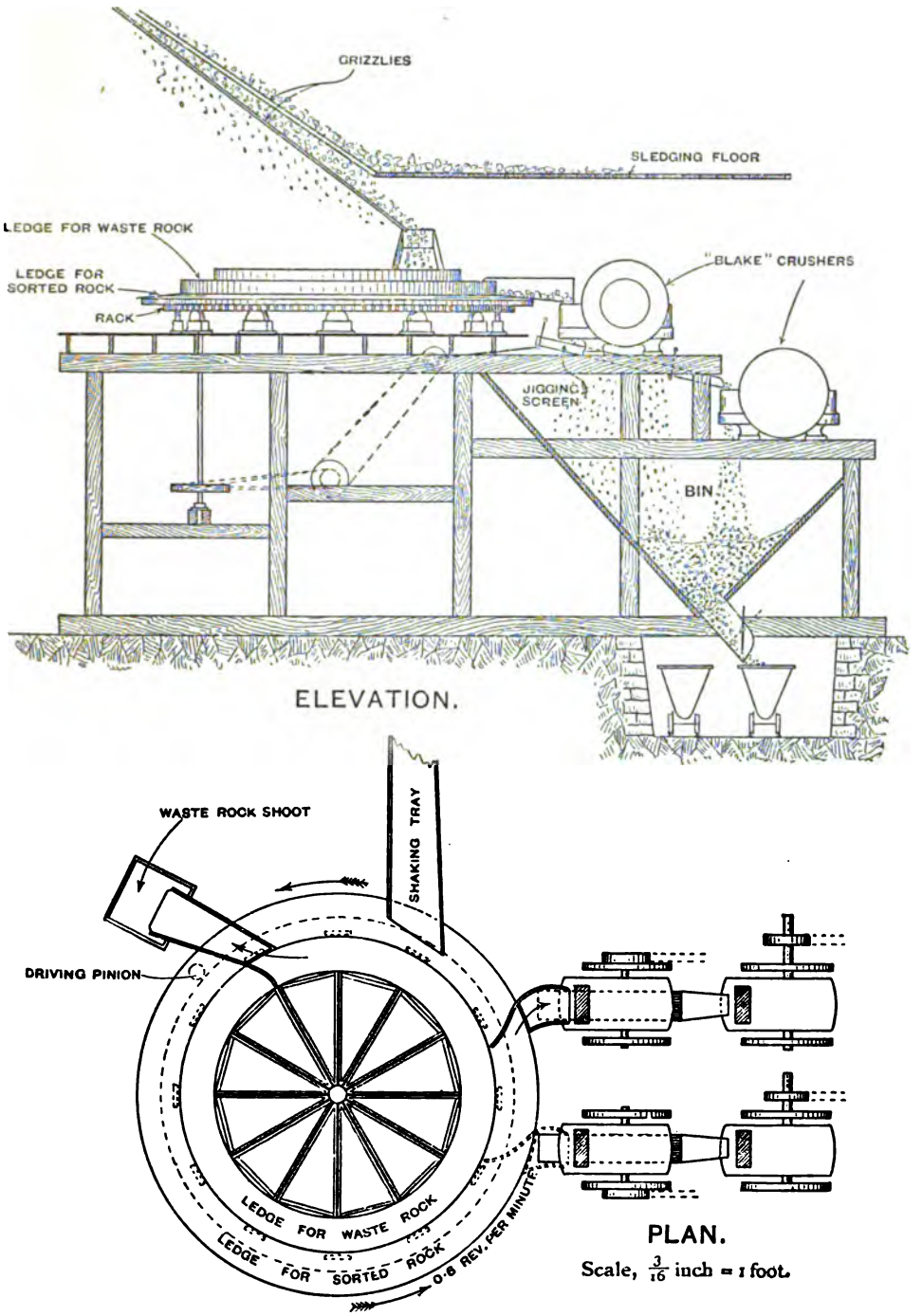


FIG. 194.—24 foot circular sorting table at the Consolidated Main Reef Mines and Estate, Limited.

machinery, the sorting belt is endless. It travels round two pulleys, situated from 35 to 40 feet apart, through one of which the motion is imparted, and between which both the upper and lower surfaces are carried on carrier pulleys. It is generally set at an angle of 10° to 15°, so that one end is higher than the other, though sometimes it is horizontal. The usual width is about 2½ feet, and the outside edges are turned up about 2 inches in order to keep the ore from falling off along the sides.

The lower end is situated just at the bottom of a grizzly and the upper end is immediately over the crushers. The ore after having been screened falls on to the upper surface of the belt, and is cleaned at once by water from a hose or from fixed jets. As it travels towards the crushers the waste is removed and dropped through shoots into bins, and at the end pulley the sorted ore drops on to the crusher floor or into the crushers. The larger pieces containing ore and waste are taken off and broken up with hammers on a floor.

There are three kinds of belts at present working. One consists of iron plates, each the full width of the belt, hinged together and working round two hexagonal pulleys; each plate carries a similar-sized tray with the outside edges turned up, on which the ore is carried; this style of belt is in use at the New Primrose, where one lasts about seven months.

Another, Greenings belt, is made up of a large number of links; such an one is in use at the Meyer and Charlton; and the third is made up of wire ropes, which are kept in line with one another, by bands of iron placed at short intervals across the width. Such an one is in use at the New Kleinfontein.

Sorting on a Shaking Table.—The shaking table is an old arrangement by which ore received irregularly at one end of a shoot is fed away more or less regularly at the other. It consists of an inclined table so suspended that after it has been displaced in a longitudinal direction it returns to its

position, and in doing so receives a knock, the return motion being effected either by gravity or with the aid of a spring.

The shaking table for sorting is placed so that its upper end receives the ore from the bottom of the grizzly, and its

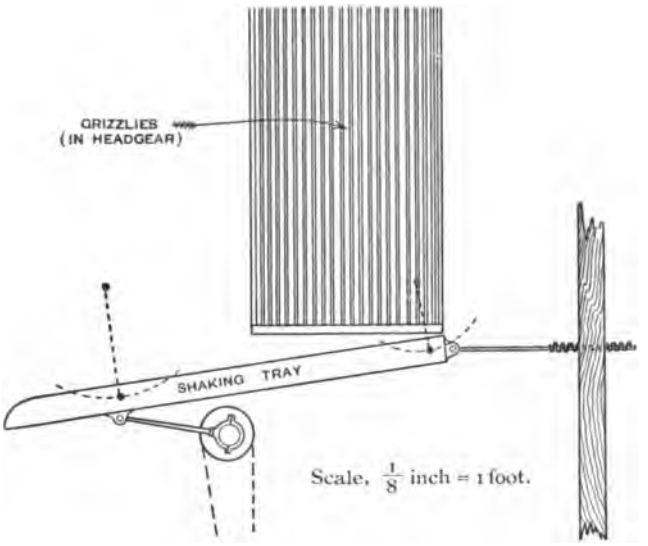


FIG. 195.—Shaking tray or table.

lower end delivers it to the crushers. At an early stage of its progress it is cleaned with water, after which the waste is picked out. The general arrangement of such a shaking tray is shown in Fig. 195.

Comparison of Methods.—The sorting floor is in extensive use on the Ferreira, Princess Estate, Geldenhuis Estate and Jumpers mines. The plants of the first two mines are very similar, and one—that of the Ferreira—has been described. On these two mines about 33 per cent of the ore mined is sorted out, of which the assay value is satisfactorily low, that of the Ferreira averaging less than 1 dwt. The advantages of a sorting floor are that with a large screening surface there is no congestion—the ore can be tipped where desired; and secondly, that no power is required to run it, in consequence of which the wear and tear and maintenance cost are low. It is stated that a disadvantage in using the floor is that the

sorted ore has to be handled in addition to the waste, but as it follows from this that every piece of rock is turned over, this would rather appear to be an advantage.

Of the other methods, by all of which the waste rock only, and not the ore, is handled during sorting, the circular table with two shelves would appear to be the best, because the ore and the waste are each taken round by the table to be deposited in their respective bins, and whilst the waste is upon the upper shelf, it is open for inspection by the overseer. It is in use on the Wemmer, Treasury, and other mines. At the first-named mine about 28 per cent of the rock mined is sorted out, and a somewhat higher result is anticipated for the second. In addition to these, circular tables are in use at several mines, including the Crown Deep, Geldenhuis Deep, City and Surburban, and provision is being made for their adoption by other mines. The wear and tear in connection with them is not great.

The working costs at the Crown Reef for April, May, and June 1897 averaged 2s. 0.655d. per ton of rock sorted out, and the amount sorted out averaged 15.21 per cent. At the City and Surburban, where about 21 per cent is sorted, the average cost is about 9d. per ton sorted out.

Sorting belts do not offer a great sorting area; the wear and tear is heavy and the cost of maintenance consequently high, but a point in their favour is that the ore may be raised some few feet whilst the sorting is going on.

Shaking tables present a small area for the display of the rock, which in consequence often lies deeply on them, so that it is likely that many pieces may pass into the crusher without having been subjected to an adequate examination.

In conclusion, it is noteworthy that at present the largest percentage sorted out is being done on sorting floors at a low cost.

The sorting floor, when at the crusher station, is best arranged half on either side of a row of Gates crushers, as at

the Jumpers mine, for then the sorted ore can be shovelled directly into the crushers.

For sorting the reef cut during development work, a sorting floor is best adapted because when no power is available in the early stage of a mine, such a floor can be easily arranged on a suitably small scale.

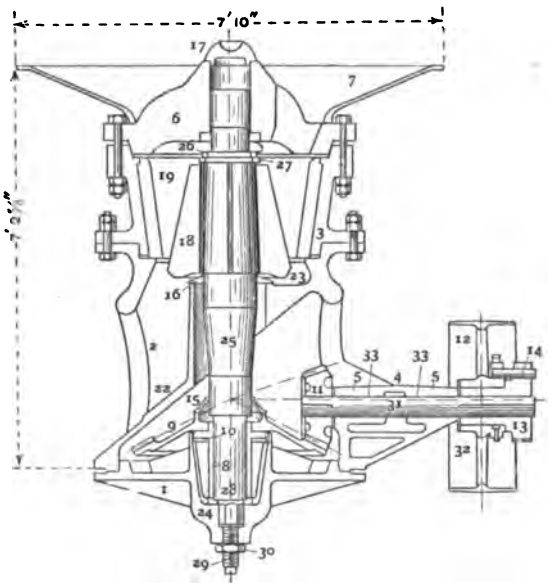


Fig.	Name of Part.	Fig.	Name of Part.	Fig.	Name of Part.	Fig.	Name of Part.
1.	Bottom Plate.	10.	Wearing Ring.	19.	Concaves.	27.	Lower Ring Nut.
2.	Bottom Shell.	11.	Bevel Pinion.	20.	Key Concaves.	28.	Steel Point.
3.	Top Shell.	12.	Driving Pulley.	21.	Narrow Concaves.	29.	Lighter Screw.
4.	Bearing Cap.	13.	Break Pin Hub.	22.	Chilled Wearing Plates.	30.	Lighter Nut.
5.	Oil Cellar Cap.	14.	Break Pin.	23.	Auxiliary Diaphragm.	31.	Counter Shaft.
6.	Spider.	15.	Oil Bonnet.	24.	Octagon Step.	32.	Driving Pulley Wearing Cotter.
7.	Hopper.	16.	Dust Ring.	25.	Shaft.	33.	Oiling Chain.
8.	Eccentric.	17.	Dust Cap.	26.	Upper Ring Nut.	34.	Babb't'g Sleeves.
9.	Bevel Wheel.	18.	Head.			35.	Babb't'g Mandrel.

FIG. 196.—Gates Crusher. Size No. 5.

Crushing.—Crushing of the ore is effected chiefly by that class of gyrating crushers of which the "Gates" crusher is a type, though a good deal is also done by the reciprocating or "Blake" type of crusher.

Gates Crusher.—A sectional elevation of this crusher, with the names of the various parts, is shown in Fig. 196.

The crushing is done between a cone carried by a gyrating

shaft which passes vertically through the centre of a conical shell tapering downwards. As it gyrates the crushing cone impinges against the sides of the shell in relation to which it is constantly approaching and receding. The top of the shaft carrying the crushing cone is held rigidly while the bottom is gyrated (with a certain amount of eccentricity depending upon the size of the machine) by means of simple gearing.

These machines are made in several sizes. The following are some particulars of the two, Nos. 3 and 5, which are in greatest use :—

	No. 3.	No. 5.
Width of annular receiving opening .	7 inches.	10 inches.
Weight of crushers	14,000 lbs.	29,000 lbs.
Size to which the product can be crushed	$1\frac{1}{4}$ inch.	$1\frac{3}{4}$ inch.

For ordinary use the head is made of chilled white iron, but for harder rock it is of manganese steel.

This machine is arranged so that when placed on timber or masonry foundations the bottom plate with the eccentric box and gear wheel may be taken out from below for repairs, and at the same time the working portion of the breaker is exposed.

The Blake Crusher.—A sectional view of this crusher is shown in Fig. 197, in which the segment marked R is the pulley, which may run in either direction ; F is the pitman, or connecting rod, operated by an eccentric; L, L are toggle plates, which, as the pitman is raised and lowered with each revolution of the shaft, impart reciprocating motion to the movable or swinging jaw, D ; H is the shaft from which the swinging jaw, D, is suspended ; E, E, E, E are steel bearings. These are lubricated by means of the oiling tubes, P, P, P, P. A tension rod, M, connects the swinging jaw with a rubber spring, which ensures the rapid return of the jaw, D, after the material has been broken. A check plate, S, keeps in its place the fixed jaw, B, against which the material is broken.

Both jaws are provided with renewable steel on chilled cast-iron plates, C. The width between the jaws is regulated by means of the wedge, K, which is set by the screw, O. The entire frame is cast in one piece.

The crushing is done by the advance of the swinging jaw against the stationary jaw.

The size which is most generally used is that which has a receiving aperture 10 inches long by 20 inches deep, though

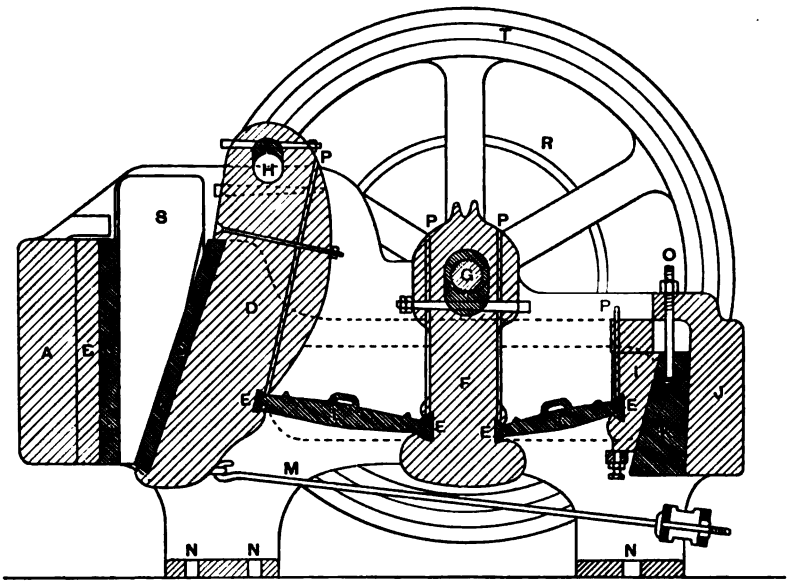


FIG. 197.—Blake Crusher.

sometimes these measurements are as much as 20 inches and 24 inches respectively.

Generally the ore is broken down to at least a 2-inch cube, independent of the class of crusher used, and this has been found to help the mill considerably; it is only when the ore is so broken, that automatic feeders for the mortar-boxes can be successfully employed. When it is desired to crush finer than this, say to $\frac{3}{4}$-inch cube, it is better to do it in two stages, such crushing being called "tandem crushing," for then the amount of fines produced is somewhat less and the crushers are not overtaxed.

The Gates crusher at all times receives the ore better than the Blake, and it is especially useful in connection with the sorting floor, for it can be fed by shovelling from all points around its circumference.

The Blake crusher, in consequence of its reciprocatory movement, sets up considerable vibration in its supports, which have in consequence to be specially stayed.

At the City and Suburban, where Gates crushers are used, the cost is about 13d. per ton actually crushed, equivalent to 6¾d. per ton milled, and at the Ferreira with Blake crushers it is just under 1s. per ton actually crushed, or 5½d. per ton milled.

The great item of cost in using the Gates crusher is that of maintenance and repairs, but latterly, by simplifying the machine and making it more convenient to operate, this has been somewhat lessened.

For an equipment of 100 stamps crushing blue rock, three of Gates No. 5 crushers are generally found necessary, one of which is kept in reserve to be used when either of the others is undergoing repair.

In all cases the crushers are fixed over ore bins which are generally of considerable capacity, and from which the ore is taken to the mill, where the operations and processes, which are included under the inclusive term of "Reduction," commence.

CHAPTER XX[1]

ADMINISTRATION, LABOUR AND MATERIAL

Administration, etc.—The mines are, almost without exception, controlled by a board of directors consisting of shareholders whose position on such a board is confirmed by the others; of these directors, one is elected the chairman and one is often made the managing director. This board represents the shareholders or owners.

In Section II. of the Mining Regulations[2] the following interpretation of the term "owner" is fixed by law as under:—

"The 'owner' of a mine shall be taken to be any person or body of persons who is the immediate holder or lessee of any mine or part thereof, and shall not be taken to include a person or body of persons who merely receives a royalty or rent from a mine, or who is merely the owner of a mine subject to any contract for the working thereof, or who is merely the owner of the soil. A tributor for the working of any mine, or any part thereof, shall be subject to this law in like manner as if he were an owner, but so as not to exempt the person or the body of persons to whom he pays tribute from any liability as under this law.

"Where a mine is owned by a company or syndicate, the chairman of such a company or syndicate will be considered to be the owner in terms of this law."

The responsibility of the owner of a mine, as stated in Section XXI. of the Mining Regulations, is given in Appendix I., p. 470.

[1] Through the following pages it was not found possible in this Second Edition to alter the names of Government or Country.

[2] Law No. 12, 1896.

In the majority of cases the board of directors meets in Johannesburg, in which case the head office of the company is also in that town.

Where the board does not meet in Johannesburg but has its head office elsewhere, it is usual for it to be represented in Johannesburg by a local board or committee.

At the head office, the secretary with the assistance of an adequate staff carries out the various secretarial duties, such as the book-keeping, the distribution of dividends, the issue, registration and transfer of shares, and the payment of accounts.

In addition to the head office, it is usual to find that a company is represented by a secretary, agent or committee, at places where the number of shares taken up would warrant such expenditure; thus most companies are represented in London, and a good many in Paris and Berlin.

Where the mines are situated quite close to Johannesburg the head office is usually situated on the mine itself, in which case all the secretarial work is under the immediate supervision of the secretary, but in the majority of cases there is on each mine a resident secretary, who with the necessary assistance carries on such secretarial work as the payment of wages, the timekeeping and storekeeping, etc., which is of necessity done at the mine.

The board of directors is advised by a consulting engineer in conjunction with a manager. The position of the latter is defined in Section II. of the Mining Regulations in the following terms:—"The term 'manager' shall be taken to mean the person registered as responsible under this law for the control, management and direction of the mine."

The work covered by all the officials who have been mentioned, and the expenses incidental to their duties and offices, are often considered respectively as constituting and as being chargeable to "administration," though there is very great inconsistency about the range of that term. It is more usual, however, to find that such expenditure is included under

the term "general charges" together with other miscellaneous items.

The advice of consulting engineers, whether in a general, mechanical or metallurgical capacity, is sought because it cannot be expected that a manager shall be an authority in all branches of mining and metallurgy. He is generally, by experience or training, a mining rather than a metallurgical engineer, and he has to hold a certificate of competency from the Government, such certificate now being only obtainable after the regulations for the examination of mine managers given in Appendix I., p. 474, have been satisfied.

Under the manager, and connected with mining proper, there are the following employees:—Mine overseer, surface foreman, shift-boss and ganger. These are defined in the Mining Regulations as follows:—

"The term 'mine overseer' shall be taken to mean the person in charge of all underground works of a mine, acting under the direction of the manager."

"The term 'surface foreman' shall be taken to mean the person in charge of the surface works of a mine, acting under the direction of the manager."

"The term 'shift-boss' shall be taken to mean the person in charge during a shift in a section of or in all the underground works of a mine, acting under the instructions of the mine overseer."

"The term 'ganger' shall be taken to mean a person in charge of a gang of workmen in one or more working places in or at a mine, but he shall not conduct blasting operations unless he be the holder of a 'blasting certificate.'"

It is one of the duties of the administration to keep distinct all the costs in labour and material, etc., of the several operations. It is this desideratum which determines the nature of the book-keeping which has to be done.

As in the opinion of those able to judge, the books of the Crown Reef G. M. Company reflect great credit on those

responsible for them, some of the results tabulated in the Annual Report of that Company for the year ending 31st November 1897 are given here. The total expenditure for working expenses and additions to plant during that year was £267,953 : 11 : 5, which amount, on a milled tonnage basis, is distributed as under :—

	Total.			Per Ton milled.	
	£	s.	d.	s.	d.
Mining expenses	165,139	14	6	16	6.268
Transport	2,839	12	9	0	3.409
Milling	26,761	13	7	2	8.130
Cyanide	28,579	13	9	2	10.073
Slimes	5,781	3	11		6.940
General charges	29,562	0	4	2	11.492
Mine development	9,489	12	1		11.393
	£267,953	11	5	£1 6	9.705
Depreciation amounted to	15,615	8	4	1	6.748
	£283,568	19	9	£1 8	4.453

This distribution gives the main items which total up to the "working costs." The details of these items, separated out under the various operations, are given in the statement which faces this page, which is a condensation from a series of tables in the General Manager's Report, and which is particularly interesting because the maintenance costs are kept separate from the actual working costs. Another statement, showing the details of these same items separated out under the costs of wages and material, is given on pp. 434 and 435, in which the various percentages written off for depreciation are also given.

Included in the item "general charges" are the cost of the sorting floor at the Main Incline Shaft, the mechanical haulage at the Cyanide Works, new circular tank for concentrates, the expenditure to date on two 40 feet by 35 feet vats for sand treatment, the high pressure cylinder for the mill engine, new dynamite magazines and mealie-grinding mill. These items amount to £6433 : 18s. or 7.724d. per ton.

LAY	TOTALS ON TONNAGE MINED.		ORE ADDED T	
	Cost.	Cost per Ton.	Tons. (P)	
	£ s. d. 164,060 12 9	£ s. d. 0 16 6.859	929	7

age presents cost of all Timbering in the Mine, excepting Main

REA		FIRE ASSAY VALUE.			
Slime	Average.	Sands.	Concentrates.	Slimes.	S
81,5	£ s. d. 1 1 7.861	dwts. 6.073	dwts. 37.391	dwts. 4.787	d 1

EXPLANATORY NOT

e value of the Bullion is estimated monthly, according to its

	TOTAL.		Average Cost per Ton on Milled Tonnage Basis.
	Cost.	Cost per Ton.	
	£ s. d. 5781 3 11	£ s. d. 0 5 4.417	£ s. d. 0 0 6.940

of L

TOTAL REVENUE.		
Value.	Value per Ton.	Total Profit.
£ s. d. 470,694 16 11	£ s. d. 2 7 1.120	£ s. d. 202,741 5 6

The following is a further statement in which details are given of the total expenditure in labour, stores and general charges.

EXPENSES FOR FINANCIAL YEAR

1st April 1896 to 31st March 1897

199,898 Tons Milled.		Cost.	Cost per Ton.	Percentage of Total Cost.
Native labour		£64,810 5 2	£0 6 5.812	24.187
Native food		12,811 2 11	0 1 3.381	4.781
White labour, salaries, etc.		78,234 17 7	0 7 9.929	29.197
Coal		25,583 5 9	0 2 6.715	9.547
Dynamite		25,283 0 10	0 2 6.355	9.435
Cyanide		8,389 18 11	0 0 10.073	3.131
Zinc		599 19 2	0 0 0.720	0.223
Royalty		5,519 10 3	0 0 6.626	2.059
Mining timber		2,125 7 10	0 0 2.551	0.793
Timber, deals, etc.		3,370 12 5	0 0 4.046	1.257
Steel		2,708 18 1	0 0 3.252	1.010
Steel bar and sheet		812 16 2	0 0 0.975	0.303
Oils, grease, paraffin		2,369 3 0	0 0 2.844	0.884
Candles		3,185 11 9	0 0 3.824	1.188
Ropes, steel, and manilla		612 17 1	0 0 0.735	0.228
Forage, chaff, and bran		2,357 17 7	0 0 2.830	0.879
Electric spares		2,624 13 8	0 0 3.151	0.979
Mill spares, shoes, dies, cams, cam shafts, stems, mortar-boxes, screening, etc.		4,016 16 6	0 0 4.822	1.499
Trucks, wheels, rails		1,588 14 6	0 0 1.907	0.592
Pipes and pipe-fittings		2,865 4 1	0 0 3.439	1.069
Fuse and detonators		1,575 16 1	0 0 1.891	0.588
Lime		1,026 8 10	0 0 1.232	8.383
Sundry stores:—bar iron, bolts and nuts, assay chemicals, machinery, etc.		7,765 15 7	0 0 9.323	2.898
General charges:—				
Insurances	£1416 1 0			
Licences and rents	590 9 0			
Quartz account	1079 1 9			
Printing and advertising	1966 2 11			
Sundries	2663 3 0			
		7,714 17 8	0 0 9.262	2.879
		£267,953 11 5	£1 6 9.695	99.989

To Mining Expenses—

Wages, Europeans	£23,731	2	2	
Wages, Kafirs, inclusive of food	47,895	19	3	
Explosives	11,831	7	3	
Tools, stores, etc.	22,874	9	9	
Mining timber	2,451	14	11	
Workshops account	11,243	14	8	
Quartz account	1,079	1	9	
Electric light account	932	7	7	
Contractors	43,099	17	2	
				£165,13[illegible]

Cost per ton . . . £0 16 6.268

To Transport Expenses—

Wages, Europeans	£812	2	11	
Wages, Kafirs, inclusive of food	418	16	3	
Tools, stores, etc.	614	12	7	
Workshops account	994	1	0	
				2,8[illegible]

Cost per ton . . . £0 0 3.409

To Milling Expenses—

Wages, Europeans	£6,625	18	1	
Wages, Kafirs, inclusive of food	1,791	5	2	
Tools, stores, chemicals, etc.	7,547	2	4	
Fuel	8,741	18	10	
Tailings dams	455	4	1	
Electric light account	585	1	11	
Workshops	1,015	3	2	
				26,7[illegible]

Cost per ton . . . £0 2 8.130

To Cyanide Expenses—

Wages, Europeans	£3,169	1	8	
Wages, Kafirs, inclusive of food	5,662	7	7	
Tools, stores, chemicals, etc.	12,085	9	8	
Fuel	1,218	18	11	
Royalty	5,155	1	1	
Electric light account	238	0	2	
Workshops account	850	14	8	
				28,[illegible]

Cost per ton . . . £0 2 10.073

‹KS—199,898 TONS MILLED

'imes Expenses—						
ıges, Europeans	£1,399	7	10			
ıges, Kafirs, inclusive of food	474	8	5			
ols, stores, chemicals, etc.	3,106	6	4			
yalty	364	9	2			
:ctric light account	52	12	8			
›rkshops account	383	19	6			
				£5,781	3	11
Cost per ton . . . £0 0 6.940						
:neral Charges—						
aries	£5,666	17	5			
tionery, printing, advertising, postages, and elegrams	1,966	2	11			
ences and rents	89	1	0			
spital	601	6	5			
ıdry expenditure	9,045	13	7			
'ectors' and audit fees	3,220	0	0			
neral maintenance	9,454	5	7			
	£30,043	6	11			
ıs amount to credit for interest received	481	6	7			
				29,562	0	4
Cost per ton . . . £0 2 11.492						
'ine Development Account—						
;ges, Europeans	£2,205	2	9			
;ges, Kafirs, inclusive of food	564	9	6			
›losives	396	4	1			
›ls, stores, etc.	2,354	10	2			
ıing timber	285	9	1			
›rkshops account	505	18	6			
ıtractors	3,177	18	6			
				9,489	12	7
Cost per ton . . . £0 0 11.393						
:iation—						
;ervoirs and dams, 5 per cent per annum off £7949 : 12 : 9	£397	9	8			
'niture, 25 per cent per annum off £744 : 8s.	186	2	0			
ldings, 5 per cent per annum off £26,858 : 5 : 2	1,342	18	3			
chinery and Plant, 7½ per cent per annum ›ff £181,685 : 8 : 8	13,626	8	2			
ıeral Improvements, 20 per cent per annum ›ff £312 : 11 : 7	62	10	3			
				15,615	8	4
Cost per ton . . . £0 1 6.748						
				£283,568	19	9
al working expenditure per ›n £1 8 4.453						

To Mining Expenses—

Wages, Europeans	£23,731	2	2		
Wages, Kafirs, inclusive of food	47,895	19	3		
Explosives	11,831	7	3		
Tools, stores, etc.	22,874	9	9		
Mining timber	2,451	14	11		
Workshops account	11,243	14	8		
Quartz account	1,079	1	9		
Electric light account	932	7	7		
Contractors	43,099	17	2		
				£165,139	14 6
Cost per ton	£0	16	6.268		

To Transport Expenses—

Wages, Europeans	£812	2	11		
Wages, Kafirs, inclusive of food	418	16	3		
Tools, stores, etc.	614	12	7		
Workshops account	994	1	0		
				2,839	12 9
Cost per ton	£0	0	3.409		

To Milling Expenses—

Wages, Europeans	£6,625	18	1		
Wages, Kafirs, inclusive of food	1,791	5	2		
Tools, stores, chemicals, etc.	7,547	2	4		
Fuel	8,741	18	10		
Tailings dams	455	4	1		
Electric light account	585	1	11		
Workshops	1,015	3	2		
				26,761	13 7
Cost per ton	£0	2	8.130		

To Cyanide Expenses—

Wages, Europeans	£3,169	1	8		
Wages, Kafirs, inclusive of food	5,662	7	7		
Tools, stores, chemicals, etc.	12,085	9	8		
Fuel	1,218	18	11		
Royalty	5,155	1	1		
Electric light account	238	0	2		
Workshops account	850	14	8		
				28,379	13 9
Cost per ton	£0	2	10.073		

WORKS—199,898 TONS MILLED

To Slimes Expenses—						
Wages, Europeans	£1,399	7	10			
Wages, Kafirs, inclusive of food	474	8	5			
Tools, stores, chemicals, etc.	3,106	6	4			
Royalty	364	9	2			
Electric light account	52	12	8			
Workshops account	383	19	6			
				£5,781	3	11
Cost per ton . . . £0 0 6.940						
To General Charges—						
Salaries	£5,666	17	5			
Stationery, printing, advertising, postages, and telegrams	1,966	2	11			
Licences and rents	89	1	0			
Hospital	601	6	5			
Sundry expenditure	9,045	13	7			
Directors' and audit fees	3,220	0	0			
General maintenance	9,454	5	7			
	£30,043	6	11			
Less amount to credit for interest received	481	6	7			
				29,562	0	4
Cost per ton . . . £0 2 11.492						
To Mine Development Account—						
Wages, Europeans	£2,205	2	9			
Wages, Kafirs, inclusive of food	564	9	6			
Explosives	396	4	1			
Tools, stores, etc.	2,354	10	2			
Mining timber	285	9	1			
Workshops account	505	18	6			
Contractors	3,177	18	6			
				9,489	12	7
Cost per ton . . . £0 0 11.393						
Depreciation—						
Reservoirs and dams, 5 per cent per annum off £7949 : 12 : 9	£397	9	8			
Furniture, 25 per cent per annum off £744 : 8s.	186	2	0			
Buildings, 5 per cent per annum off £26,858 : 5 : 2	1,342	18	3			
Machinery and Plant, 7½ per cent per annum off £181,685 : 8 : 8	13,626	8	2			
General Improvements, 20 per cent per annum off £312 : 11 : 7	62	10	3			
				15,615	8	4
Cost per ton . . . £0 1 6.748						
				£283,568	19	9
Total working expenditure per ton £1 8 4.453						

From all these figures it is seen that the book-keeping must be very complete.

In the grouping of the different operations to form the main items of working costs, there is not that uniformity amongst the various mines which, did it exist, would make comparison of the working expenditures so much more simple.

In these accounts of the Crown Reef it is seen that "stone-crushing" is charged to mining expenses, whereas with the advent of almost universal sorting on these fields it would be more convenient to class it with sorting, under such a heading as "ore-dressing," or in any case it would appear to be better classed as a part of milling than as a part of mining.[1] Another feature of these Crown Reef accounts is that only the main shafts and main cross-cuts are charged to development, whereas it is more usual to include drifting, rising and sinking under that heading.

It will be noticed that the different expenses are charged with their respective actual maintenance costs. No expenditure was made on capital account because the mine had already received a full and adequate equipment, and it was therefore considered that no further expenditure could consequently be included under that head.

The following Directors' Report for the month of August 1897 gives the statistics of revenue and expenditure with the summary of the work done on the property of the Witwatersrand Gold Mining Company.

[1] In the General Manager's Report for the year ending 31st March 1897, "crushing and sorting" form an item to themselves, though in the Profit and Loss Account for that year they are again included with "mining."

WITWATERSRAND GOLD MINING COMPANY

Directors' Report for August 1897

Working Expenditure

	Cost.			Cost per Ton. s.	d.
Mining	£5,771	0	4	8	10.542
Pumping and hauling	733	16	2	1	1.547
Breaking and Crushing	322	8	5	0	5.953
Transport of ore	207	0	5	0	3.822
Milling	1,325	5	2	2	0.466
Cyaniding	1,073	19	7	1	7.827
General charges	868	3	7	1	4.028
Mine development redemption	3,250	0	0	5	0.000
	£13,551	13	8	20	10.185
Profit for month	3,132	17	2	4	9.837
	£16,684	10	10	25	8.022

The costs are calculated on the basis of tonnage milled.

Revenue

	Value.			Value per Ton. s.	d.
Mill gold 3511.450 ozs.	£12,729	0	0	19	6.997
Cyanide gold 1174.596 ozs.	3,955	10	10	6	1.025
	£16,684	10	10	25	8.022

Output

	Bullion. ozs.	Fine Gold. ozs.
From mill	3511.450	3052.503
From cyanide works	1174.596	982.690
	4686.046	4035.193

Reduction

90 stamps ran	27¾ days
Ore milled	13,000 tons
Yield of bullion	3511.450 ozs.
Yield per ton milled	4.402 dwts.
Duty per stamp per day	5.200 tons

Cyaniding

Tailings treated	9900 tons
Yield of bullion	1174.596 ozs.
Yield per ton treated	2.373 dwts.
Yield per ton milled	1.807 dwts.

Yield of Fine Gold per Ton Milled

By amalgamation	4.696 dwts.
By cyaniding	1.512 dwts.
	6.208 dwts.

MINE DEVELOPMENT

Number of feet driven, sunk, and risen, 743 feet; development at 31st July, 214,438 tons; ore developed during month, 23,679 tons; ore milled, 13,000 tons; total ore reserves at 31st August, 225,117 tons.

CAPITAL EXPENDITURE

Permanent works	£477	14	10
Buildings	1398	16	11
Machinery and plant	4871	0	9
Mine development	1771	3	5
Cyanide works	265	12	7
Compressor reservoir	51	7	11
	£8835	16	5

In the cyanide works a partial clean-up only has been made.

Here "pumping and hauling" are charged together, and so are "breaking and crushing." The charge for "mine development redemption" is made in accordance with the principle explained in the speech of Mr. Lionel Phillips given on p. 302. It is to be noticed that at 5s. per ton for redemption the amount redeemed was £3250 for 13,000 tons of ore, whereas the amount actually expended during the month for mine development was £1771 : 3 : 5, and 23,679 tons were developed, which makes the actual cost of developing that tonnage about 1s. 6d. per ton, so that when it is being mined a smaller charge than 5s. for development redemption could safely be made.

It will be noticed that both in this case and the next, the New Comet, the capital account is still kept open.

The report for the month of April 1897 given on p. 439 shows the method of summarising employed by the New Comet Gold Mining Company.

From this statement it is seen that mining, hauling, and development are charged together as one item of expense, that crushing, sorting, and tramming are placed together, and

NEW COMET GOLD MINING COMPANY

Summary of Operations for the Month of April 1897

Reduction

Ore milled	6639 tons of 2000 lbs.	
Yielding smelted gold	1938.36 ozs.	£7,126 17 11
Average per ton	5.84 dwts.	

Cyanide Treatment

Tailings treated	5105 tons of 2000 lbs.	
Yielding smelted gold	1174.44 ozs.	4,079 17 7
Average per ton	4.60 dwts.	
Total output for month, 3112.80 ozs., realising		£11,206 15 6
Rent of 40-stamp mill	£1000 0 0	
Sundry revenue	25 0 0	
		1,025 0 0
		£12,231 15 6

Cost per Ton (Milled)

	s.	d.	
Mining, hauling and development	20	1.30	
Crushing, sorting and tramming	1	11.79	
Milling	3	9.11	
Cyanide treatment	2	10.18	
Maintenance £1057 6 0 } proportioned over above account			
General charges 603 3 11 }			
6.639 tons at	28	8.38	9,526 16 4
Profit for month			£2,704 19 2

Summary

60-stamp mill ran	28 days
Average crushing per stamp per day	3.95 tons
Average yield gold per ton milled	9.37 dwts.
Value per ton ore milled	33s. 9.12d.
Cost per ton ore milled	28s. 8.38d.
Profit " "	5s. 0.74d.

Capital Account

Expended on main shafts	£175 1 5
Excess development	646 0 3
Machinery and plant	2639 5 5
Buildings	565 19 11
	£4026 7 0

Mine

	N. Reef.	S. Reef.
Total ore reserves on 31st March 1897	142,629	25,898
Number of feet driven, sunk and raised 843 feet, developing during April	8,919	1,449
	151,548	27,347
Tons ore milled—April	6,639	
Tons ore reserves April 30th, 1897	144,909	27,347

that general charges, in addition to maintenance, are distributed over the other expenses.

It is also to be noticed here that the excess development, *i.e.* the tonnage developed during any period in excess of that milled, is charged to capital account; this excess amounted to 3729 tons, and cost £646 : 0 : 3, or at the rate of about 3s. 5d. per ton.

With this method of treating development, where during any month more ore is developed than milled, the total cost of development is divided into two parts proportionately to the amount milled and to that developed in excess; that part which is proportionate to the amount milled is charged to working expenses and the other part is charged to capital account, the excess tonnage at the same time being placed to the credit of the mine as "ore reserves," by which means the ore reserves have a value as an asset.

When during any one month more ore has been milled than developed, then the ore reserves have to be drawn upon at a redemption cost for development, which can be calculated from the figure at which the total reserves stand in the books.

At the Geldenhuis Estate the whole cost of development is included as part of the working costs, as shown in the extract from the Directors' Report for June 1897 given on p. 441, and no charge is made for excess development.

Referring to that extract, it is seen that 40,603 tons were developed; the expenses of developing this tonnage are distributed over the 15,644 tons milled, in accordance with the principle mentioned on p. 303. In these accounts also, maintenance is a separate item, though it is perhaps better to keep the different maintenance costs for the different items distinct one from the other, so that they may be charged under their respective headings, as shown in the statement of the Crown Reef facing p. 432, where, in most of the items of expense, the working cost and the maintenance cost together make up the total cost.

GELDENHUIS ESTATE AND GOLD MINING COMPANY, JUNE 1897

Working Expenses

	Total.			Per Ton milled. s.	d.
Mining	£6,128	3	0	7	10.01
Hauling and pumping	666	19	7	0	10.23
Sorting, tramming and crushing	514	19	8	0	7.90
Development	2,035	15	11	2	7.23
Milling	1,498	19	7	1	11.00
Cyaniding concentrates	172	17	10	0	2.65
Cyaniding tailings	1,261	15	10	1	7.36
Mill-water supply	212	14	4	0	3.27
Maintenance	2,069	14	11	2	7.75
Charges	529	1	7	0	8.12
	£15,091	2	3	19	3.52
Profit for month	15,066	17	9	19	3.15
	£30,158	0	0	38	6.67

Revenue

		Total.			Per Ton milled. s.	d.
Gold from mills	6188.49 ozs. valued	£21,683	0	0	27	8.65
Gold from tailings	2023.36 „ „	6,296	15	0	8	0.60
Gold from concentrates	700.00 „ „	2,178	5	0	2	9.42
		£30,158	0	0	38	6.67

Mine Development

Total footage for month	1,573½ feet
The ore developed by the above footage was	40,603 tons

Mill

120 stamps ran 28 days 8 hours, crushing	15,644 tons
Tons crushed per stamp per 24 hours	4.60 „
Bullion yield	6188.49 ozs.
Bullion yield per ton	7.91 dwts.

Cyanide Works

Tons of tailings treated	10,115
Bullion yield	2023.36 ozs.
Bullion yield per ton	4.00 dwts.
Working cost per ton	2s. 5.9d.
Tons of concentrates treated	1120
Bullion yield	700.00 ozs.
Bullion yield per ton	12.50 dwts.
Working cost per ton	3s. 1.05d. per ton

In the Geldenhuis Estate accounts it is also noticeable that several other items of expense are separated out, which in other accounts would be embodied under more important headings. Thus the mill water-supply is generally included with milling; the cyaniding of concentrates is generally included under cyanide costs. In these accounts sorting, tramming, and crushing are kept together, as was also the case with the New Comet.

It will be noticed that in these monthly reports no depreciation is charged; it is the practice to charge for this at the end of the financial year, the amount being at the discretion of the directors. This depreciation when distributed over the tonnage milled and included as working costs makes these costs in the annual report somewhat greater than those appearing in the directors' monthly reports; thus Mr. Hennen Jennings stated, in his evidence before the Industrial Commission, that from the annual reports of twenty-nine of the principal companies working in 1896 the total working cost without depreciation was 27s. 4d. per ton, and with depreciation 31s. 0.44d. per ton, making for depreciation alone 3s. 8.44d. Some mines, however, record the amount put down for depreciation as a book entry which is not to be considered a part of working costs, but which is subtracted from the profits, to redeem capital or to be spent as capital would be. These twenty-nine companies, during the period covered, crushed 3,444,482 tons, from which the value of the total recovery of gold was £7,008,265, of which £1,761,103 were distributed as dividends. If we subtract this latter sum from the former and then consider the balance of £5,247,162 as having been spent for working costs, then the working cost per ton works out at 30s. 5½d., or very nearly the same as that which includes depreciation.

For all the producing mines which were running, the following yearly average working costs, calculated out by this method, were obtained.

	s.	d.
In 1894	38	10.00
„ 1895	33	5.25
„ 1896	31	6.90
„ 1897	29	6.70

This table shows the progress in the perfection of methods and machinery which is being made in mining and extraction, especially when it is considered that during 1894 the extra cost due to sorting, and the apparent extra total cost due to the distribution of the whole expenses over the tonnage milled, and not over the larger tonnage mined, were not incurred, whereas during 1897 they did occur and in addition there was in that year some further expense due to the introduction of the slimes process.

If all fully-equipped mines were to consider all current expenditure, including that on improvements in equipment, etc., as working costs, then the average cost per ton milled would be somewhat greater. In 1897 some few companies did this, but in 1894 it was not done.

The mines in the Central, Van Ryn-Chimes, Krugersdorp and Heidelberg districts, which are generally included as the Witwatersrand district, incorporated themselves into a body called "The Chamber[1] of Mines of the South African Republic"; this chamber was recognised by the late Government as the embodiment of the mining industry. The mines themselves, through proper representatives, are the members of this chamber, which consequently is vitally interested in promoting and protecting the industry; the statistics which are thus collected are most important, and they form one of the most valuable products of this chamber. Every month a monthly analysis of the gold production is tabulated in the manner shown by the blank specimen facing p. 444 and issued, and at the end of the year a full report

[1] Since the commencement of 1898 all the gold mining companies in the outlying districts of Lydenburg, De Kaap, and Klerksdorp declare their output to this Chamber.

with the annual statistics are adopted at a meeting, when it is usual for the President of the Chamber to give a review of the last year's work. It is from the President's annual speech on 20th January 1898 that the following statistics showing the chief features of the work for the year 1897 were mainly taken. During 1897, 3,034,678 ozs. of gold bullion, representing a value of £10,583,616, were returned from the mines of the Chamber, which is an increase over the year 1896 of 753,785 ozs. and £2,719,275. This amount of gold was won from the different operations in the following proportions:—

	Per cent.
Mills	64
Tailings	34
Concentrates	2
	100

The mills crushed 32.7 per cent more than in 1896, and the value of the gold recovered from them was 32.4 per cent more.

The cyanide works treated 35 per cent more, and the value of the gold recovered from them increased 45.1 per cent.

The value of the gold returned from the concentrates was 17.6 greater than in 1896.

During 1897 the average number of companies milling per month was fifty-one, with an average total of 3567 stamps against forty-two companies, and 2949 stamps in 1896; the tonnage crushed in 1897 was 5,325,355 tons, an increase of 1,313, 658 over 1896.

Altogether sixty-nine companies contributed towards the total output in 1897 and fifty-nine in 1896.

The duty per stamp in 1897 was 4.6 tons crushed per day of twenty-four hours and 4.4 tons in 1896.

In 1896 the average value of the yield from all sources was 39s. 2d., and in 1897 39s. 9d.[1]

Leaving out the Bonanza, five deep levels contributed

[1] In 1895 the average value of the yield was 45s. 4d., of which 11s. 10¾d. was distributed in dividends.

O

GOLI

To face page 444.

N	No.	Pl	Total value of yield per ton milled.	Concentrates produced during the month. Tonnage produced.	Assay Value per ton in ozs.	No.	REMARKS.
			s.				
							General Summary.
							ozs. dwts. Value.
							Witwatersrand . . .
							Lydenburg . . .
							De Kaap . . .
							Klerksdorp . . .
							Grand Total . .

:erned.

6.4 per cent of the total output of the year, although the majority of them put in but a small portion of a year's work. The capacity of the deep levels is better illustrated by the fact that during December 1897 four deep levels contributed 12.6 per cent of the total output for that month.

The total amount distributed in dividends during the year by the companies contributing to the output was £2,713,580, equal to 25.64 per cent of the output or to 10s. 2.3d. per ton milled.[1]

Most of these figures are shown in the tabulated statement facing p. 446, which is the annual analysis of the gold production for 1897.

The progressive nature of the output from the Witwatersrand goldfields is shown by the following figures :—

In 1893	the output was	1,478,473	ozs.
„ 1894	„	2,024,159	„
„ 1895	„	2,227,640	„
„ 1896	„	2,280,892	„
„ 1897	„	3,034,678[2]	„

These figures include the outputs from all the districts where banket is being worked, with the exception of the Klerksdorp district; they do not include the other outlying districts of Lydenburg and De Kaap.

There exists in the South African Republic a Department of Mines at the head of which is the Minister of Mines. The Minister of Mines is supported by a qualified expert, called the State Mining Engineer, who is assisted by mining inspectors and boiler inspectors on the fields. These officials act in accordance with a Gold Law and with Mining Regulations. Their annual reports give valuable statistics relative to the entire mining industry of the Republic, including coal as well as gold. From the report of the State Mining

[1] In 1896 the dividends distributed amounted to 7s. 1d. per ton milled.

[2] For the first half of the year 1898 the output of the Witwatersrand district together with that of the outlying districts was 2,088,808 ozs. of gold bullion, neglecting the odd dwts. each month.

Engineer, the following figures of the value of the total gold output of the whole country were taken :—

				£
In 1884	the value	of the	output was	10,096
„ 1885	„	„	„	6,010
„ 1886	„	„	„	34,710
„ 1887	„	„	„	169,401
„ 1888	„	„	„	967,416
„ 1889	„	„	„	1,490,568
„ 1890	„	„	„	1,869,645
„ 1891	„	„	„	2,924,305
„ 1892	„	„	„	4,541,671
„ 1893	„	„	„	5,480,498
„ 1894	„	„	„	7,667,152
„ 1895	„	„	„	8,569,555
„ 1896	„	„	„	8,603,821
„ 1897	„	„	„	11,653,725

Of these figures the Witwatersrand goldfields produced over 91 per cent. In 1897 there were 10,002 whites and 75,788 natives employed on the coal and gold mines against 9818 whites and 69,657 natives in 1896. Amongst the most important of the returns of the State Mining Engineer are those relative to accidents. During the year 1896, at the mines and in connection with the boilers in use in the South African Republic, there were 648 accidents, 389 of which were fatal, and in 1897 there were 634 accidents, of which 369 were fatal—47 white and 222 natives.

In round figures the following percentages are the proportions of accidents due to the various causes :—

	Per Cent.
Missed fire charges (blasting)	15.00
Other dynamite explosions	12.50
Machinery and boilers	7.75
Falls of ground	18.00
Falling materials	6.50
Truck and tramway transport	6.00
Carriage by cage or skip	18.25
„ ladder	1.00
Falls in shaft	7.50
Miscellaneous causes	7.50
	100.00

1897

To face page 446.

COMPANY.	Tons Tons 2000	LINGS. OF GOLD.		VALUE OF YIELD.				FROM ALL SOURCES.			REMARKS.
			Per Ton Bullion.	Total.	Per Ton.			Total Yield.		Total Value.	
		ns.	dwts.	£	£	s.	d.	ozs.	dwts.	£	
Angelo . . .	64,	18	8.14	67,608	1	8	2	52,491	16	188,943	10 months' work
Balmoral . . .	30,	10	3.12	13,143	0	11	11	10,986	6	40,647	6 months' work
Banket . . .	4,	7	3.94	1,961	0	9	7	2,418	2	7,895	4 months' work
Bonanza . . .	63,	9	13.66	114,303	2	7	9	93,107	17	325,354	9 months' work
Champ d'Or . .	36,	15	5.52	22,046	0	16	10	23,221	16	79,360	
City and Suburban .	226,	4	4.82	143,920	0	16	1	133,673	8	483,413	
Crown Deep . .	71,	18	7.96	74,879	1	6	4	48,032	11	162,508	5 months' work
Crown Reef . .	188,	4	5.19	137,810	0	16	7	148,946	6	504,309	
Durban Roodepoort .	122,	10	3.93	53,275	0	12	10	71,177	10	260,352	
Ferreira . . .	125,	6	6.94	108,221	1	2	7	131,363	5	459,828	
Geldenhuis Deep .	208,	14	5.10	133,561	0	16	5	108,663	2	377,029	
Geldenhuis Estate .	194,	11	3.86	74,589	0	11	10	104,943	9	351,275	
Geldenhuis Main Reef	39,	11	5.71	21,504	0	17	1	19,461	13	64,558	
George Goch . .	108,	2	3.85	43,897	0	11	6	36,465	8	122,997	
Ginsberg . . .	61,	2	5.31	44,997	1	0	2	36,440	1	135,130	
Glencairn . .	84,	18	4.14	38,057	0	14	2	32,453	8	117,734	6 months' work
Henry Nourse . .	101,	10	7.75	86,966	1	3	3	91,651	5	314,056	
Johannesburg Pioneer	36,	17	8.90	42,264	1	6	6	43,617	2	148,020	
Jubilee . . .	59,	7	4.80	27,134	0	15	6	30,903	14	110,773	
Jumpers . . .	137,	14	2.87	38,835	0	9	6	61,809	1	221,359	
Lancaster . .	9,		...	...		...		3,325	8	12,238	2 months' work
Langlaagte Block "B"	119,	9	2.95	28,799	0	8	10	44,871	19	150,255	
Langlaagte Estate .	305,	12	2.67	72,800	0	8	0	128,848	18	429,450	
Langlaagte Royal .	..	0	3.43	4,885	0	11	1	1,857	0	6,175	Clean up
Langlaagte Star .	10,	15	4.81	5,740	0	14	5	7,945	15	27,133	2 months' work
Langlaagte United .	45,	12	4.63	22,131	0	14	10	17,860	1	63,679	7 months' work
Marie Louise . .	4,		...	...		...		1,858	7	5,558	Dry crushing, 3 months' work
May Consolidated .	132,	0	4.42	65,148	0	13	3	59,343	6	201,567	
Meyer and Charlton .	109,	15	4.15	44,891	0	12	5	46,080	6	159,444	
Minerva . . .	25,	0	4.85	6,714	0	11	1	5,698	8	17,010	7 months' work
New Comet . .	69,	17	5.41	49,285	0	18	5	37,709	7	134,502	
New Hidlbrg Roodept	32,	13	3.01	11,397	0	8	0	9,727	6	29,719	8 months' work
New Heriot . .	109,	8	7.68	84,995	1	3	3	72,908	11	249,921	
New Kleinfontein .	102,	15	2.87	29,761	0	8	6	43,573	17	147,544	9 months' work
New Midas . .	43,	19	4.71	16,502	0	14	1	12,034	3	39,390	
New Modderfontein .	77,	16	3.24	26,602	0	10	6	30,440	12	107,735	Shut down during December
New Primrose . .	278,	3	5.26	151,434	0	17	4	116,262	7	408,767	
New Rietfontein .	11,	0	3.19	5,037	0	9	5	4,914	5	16,769	3 months' work
New Spes Bona .	60,	14	5.13	32,589	0	16	10	23,777	6	82,405	
Nigel . . .	14,	15	10.36	18,496	1	11	0	11,843	0	37,821	5 months' work
North Randfontein .	66,	5	2.99	22,903	0	8	11	25,403	5	85,107	11 months' work
Nourse Deep . .	9,	8	9.32	7,990	1	13	7	6,099	4	21,967	1 month's work
Paarl Central . .	80,	10	4.41	36,892	0	12	8	35,709	7	119,000	
Porges Randfontein .	96,	3	2.88	28,742	0	8	7	48,998	7	166,173	
Princess Estate .	57,	0	4.54	30,182	0	13	8	34,768	1	121,925	
Rand Nigel . .	9,	10	6.73	6,426	1	0	2	5,116	10	16,997	5 months' work
Rietfontein "A" .	37,	17	5.04	19,747	0	14	9	27,425	7	97,132	6 months' work
Robinson . .	180,	5	5.22	166,953	0	17	4	189,450	5	681,197	
Roodepoort (Kmbrly)	39,	19	2.49	12,283	0	7	4	14,621	12	50,909	
Robinson Randfontein	9,	0	3.80	4,620	0	11	4	6,141	17	20,857	2 months' work [from clean up
Roodepoort Deep .		0	11.17	1,201		...		2,404	2	7,882	1 month's work, includes 1394 ozs.
Rose Deep . .	38,	8	6.48	29,560	1	0	10	24,253	18	83,518	3 months' work
Salisbury . . .	59,	4	3.05	15,172	0	9	4	23,510	18	84,645	
Simmer and Jack .	164,	1	5.37	75,881	0	16	1	75,265	19	258,098	
Stanhope . . .	28,	7	3.80	11,579	0	11	3	12,318	16	41,244	
Treasury . . .	70,	15	5.72	41,035	0	16	7	39,727	12	132,505	
The Rip . . .	1,	10	9.10	34,150	1	7	3	11,595	10	34,826	9 months' work dry crushing
United Main Reef .	86,	0	4.00	34,408	0	11	11	49,802	15	177,983	
Van Ryn . . .	130,	12	3.00	40,523	0	9	0	46,730	18	157,633	
Van Ryn, West .	68,	4	2.43	15,443	0	7	3	22,860	3	80,903	9 months' work
Violet . . .	11,	17	1.48	1,096	0	3	0	2,927	17	9,572	2 months' work
Vogelstruis . .	52,	0	1.84	10,438	0	6	0	16,765	0	61,596	6 months' work, shut down in June
Wemmer . . .	78,	15	4.39	36,258	0	14	1	58,054	1	208,212	
West Rand . .	35,	2	4.11	17,704	0	12	5	14,441	3	49,286	9 months' work
Windsor . . .	10,	2	4.73	6,384	0	14	4	4,798	2	16,447	3 months' work
Wtwatrsrnd (Knights)	154,	3	3.28	58,737	0	11	11	59,115	12	214,786	
Wolhuter . . .	160,	18	5.84	101,053	0	17	9	89,773	2	310,353	
Worcester . .	56,		...	...		...		22,117	10	78,134	Tailings sold
York . . .	3,	0	5.43	2,480	0	16	9	1,849	0	6,297	1 month's work
CUSTOM WORKS—											
Rnd Cntral Ore R. Co.	...	19	10.64	151,107	1	18	8	52,400	13	190,560	
Robinson . . .	...		...	...		...		40,244	11	167,964	
Transv'l Chemical Co.	...		...	...		...		4,460	18	17,761	
Received by Banks & from other sources	...		...	...		...		2,820	3	9,525	
	5,325,	19	4.94	2,987,123	0	15	8	3,035,678	11	10,583,616	

Ground Tenure.—Ground for mining purposes is held under the Gold Law of the South African Republic, either under "mjnpacht brief," *i.e.* mining lease, such areas being known as "mjnpachten," or under claim licences, such areas being known as "claims."

In the territory which has throughout been considered to be embraced by the term Witwatersrand Goldfields, the total area held as claims is about six times that held as mjnpachten.

A "mjnpacht" (plur. mjnpachten) is that portion of a man's piece of ground, the mining rights of which were reserved for him at the time when the remainder of the ground was thrown open by the Government as public diggings.

As stated before, they are worked under "mjnpacht brief" or mining lease, such lease being, in each case, issued for one-tenth of the original area of the piece of ground. In marking off such an area the length along the reef must not be in greater proportion to the breadth than two to one.

These leases are granted for not less than five years and not more than twenty years, and at their expiration they are renewable, subject to the then existing law. For these areas dues are paid to the Government at the rate of 10s. per morgen; in lieu of this payment, however, the Government has the right to claim 2½ per cent of the gross value of the finds.

Of the larger area, in the territory under consideration, held as claims, 96 per cent is held under prospecting licences, and the remaining 4 per cent under diggers' licences. The extent of a quartz reef claim according to the Gold Law has been given on p. 1. In addition there are the following claims, the measurements being in Cape feet:—

An alluvial claim, 150 feet by 150 feet square; a precious stone claim, 30 feet by 30 feet square; a base-metal claim, 150 feet by 400 feet rectangular.

A claim is held under diggers' licences when machinery

has been erected on it, or when ore from that claim is being crushed. Under diggers' licence 20s. per month is the full amount payable, whereas under prospecting licence only 5s. per month is payable on claims situated on private proclaimed ground, and 2s. 6d. per month on Government ground.

As stated on p. 2, all boundaries extend vertically downwards.

Claims are amalgamated into blocks at the instance of neighbouring owners, or they can be purchased from the owners, so that one person or company may become the possessor of a number of claims.

Labour.—On an average there is now one white man employed on the mines to every eight or ten natives, though the proportion of white men is increasing.

Mr. Hennen Jennings, in his evidence before the Industrial Commission, stated that for six of the principal companies during 1896

The cost of native wages was	23.73	per cent	of the total	working costs.
" " food "	4.14	"	"	"
" white labour and salaries was	30.18	"	"	"
Total percentage for labour	58.05	"	"	"

These figures may be taken as representative of the producing mines.

Mr. L. I. Seymour, in his evidence before the Commission, stated that for construction work alone the expenses are made up as follows:—

Material	$65\frac{2}{3}$	per cent
Labour	$29\frac{2}{3}$	"
General Expenses . . .	$4\frac{2}{3}$	"
	100	"

He also stated that for similar construction work in England the average cost for labour is about 31.7 per cent. During the year 1896 the statistics for fifty-six companies

showed that one quarter of a ton was mined and treated per man, white and black together, employed per shift. In these mines the average size of the stopes is about 5 feet.

White Labour.—The ordinary white miner receives about fifteen shillings per shift; there are generally six day shifts and six night shifts underground in a week in any producing mine, as no work is undertaken on a Sunday other than that which is necessary to keep the mine open and in repair. Each shift consists of from 9½ to 10 hours' work, excluding any time the miner is away from work for meals; on Saturday, however, it is usual for both the day and the night shift to work short hours, so that the night shift may be over before midnight; on other days the ordinary hours for the day shift are from 7 A.M. to 12 A.M. and from 1 P.M. to 5.30 or 6 P.M., and for the night shift from 7 P.M. to 5 A.M. In sinking the shafts of the deep-level companies it is usual to work with three shifts of eight hours each during the day of twenty-four hours, and permission is also obtained to work on Sundays.

The mechanic and carpenter are as a rule more highly paid than the miner, the usual rate for them being twenty shillings per shift. The foremen both on surface and underground are more often paid a monthly salary than a daily wage.

The following is a list of the average monthly wages for various occupations at the mines of the late South African Republic for the month of December 1896, taken from the State Mining Engineer's Report for 1896:—

	Per Month.		
Managers	£92	4	0
Mine overseers	38	11	10
Battery managers	40	5	8
Other overseers	23	10	3
Engineers	38	18	0
Surveyors	30	6	0
Draughtsmen	29	0	6
Electricians	26	9	4
Secretaries	33	1	2

	Per Month.		
Mine clerks and storekeepers	£22	10	3
Gangers	27	1	0
Miners and trammers	22	9	10
Machine drillers	27	9	5
Pitmen	18	7	9
Engine-drivers	25	2	9
Greasers	18	15	4
Pump-men	25	2	9
Stokers	18	15	3
Carpenters	25	16	10
Smiths	26	13	5
Fitters and mechanics	26	9	5
Painters	21	16	0
Masons	26	7	2
Labourers	18	1	8
Assayers	28	3	6
Amalgamators	22	7	3
Cyanide workers	23	6	10
Concentrate workers	21	16	0
Vanner workers	19	10	0
Smelters	27	16	10
Sundry workers	20	14	2

The total number of white men working in these various occupations during that month was 11,627, and they received £306,842, or at the rate of £26 : 7 : 9 per man.[1]

In developing those deep levels which are subsidiary to the Rand Mines, Limited, the average daily wage for a white man was calculated to be 19s. 6.6d. When a mine is in the producing stage it is less; for instance, at the Crown Reef it was calculated to be 18s. 6d. per day.

These different rates of pay would appear to be about 15 to 30 per cent higher than in most American mining centres. The following figures, taken principally from Mr. Seymour's evidence before the Industrial Commission at Johannesburg, are a comparison between the wages in Johannesburg, in America, and in Great Britain :—

[1] During the year 1896, according to the State Mining Engineer, the average amount paid for white labour was £317 per man, and during 1897 £311.

		s.	d.
Crown Reef (producing mine),	average wage per shift	18	6
Jumpers Deep (developing mine)	,, ,,	19	4
Montana, U.S.A.	,, ,,	14	4
Nevada, U.S.A.	,, ,,	17	3
British Columbia	,, ,,	20	6
Alston, Cumberland, England	,, ,,	4	0
Coal districts, Great Britain	,, ,,	5	0

Although the white wages are from 15 to 30 per cent higher on the Witwatersrand than in most mining centres of America, and very considerably higher than in Europe, the cost of living is more than proportionately high, and comforts experienced elsewhere are to a great extent wanting. This high cost of living bears especially upon the married man, for with a wife and two children it has been estimated that the cost of the necessaries of life will amount to £19 per month, which leaves little for life insurance, education, tobacco, amusements, medical attendance, newspapers, books, furniture, etc. In consequence, the greater portion of the married men have their families out of the country, in England. Mr. Goldmann stated in his evidence before the Industrial Commission that out of a total of 3620 white men employed

470 or 12.9 per cent were married and living with their families on the mines.
1195 ,, 33.1 ,, were married, with their families at home.
1955 ,, 54.0 ,, were single.

There are on the mines boarding-houses, where complete board costs from £5 to £6 monthly per man, so that the single man should be able to save some of his wages.

The companies in all cases, with the exception of those mines which are near to Johannesburg or any other township, provide quarters for their employees. Two men are generally placed in one room, for which accommodation, and for the further purpose of covering the cost of sanitation and cleaning, these men are sometimes charged ten shillings per month each.

For the married employees who have their families with them, small four-roomed cottages are usually provided, for which a small rent is charged. The demand for these cottages is always greater than the supply. These rooms and cottages are now often lit throughout with electricity, otherwise too many candles from the mine are burnt in them.

The wages which have been given so far refer only to day wages; the work done in driving, rising, sinking and stoping is often paid for by results under the two systems known as the "contract" and the "bonus" systems.

In working on contract, the miner and management agree upon the price of a unit of work, which price is fixed only to cover the breaking of the rock and not to include the shifting of the rock so broken, except perhaps just sufficient to clear the face that work may proceed again, or, in the stopes, to deliver the broken reef into the stope boxes.

The unit of work in all headings is the linear "foot" advanced, and in stopes the "square fathom" of area along the reef plane, though payment for stoping is in one or two cases made on the number of trucks filled with ore. All stores on which waste may occur are charged up to the miner, as, for instance, dynamite, candles, lubricants, etc.

Contract work is further referred to in the index.

In the bonus system the men receive some payment for those results which are in excess of the normal results obtained by day wages. Such normal in each particular heading is determined from experience; for every unit of work above that normal amount the miner is paid a sum which is mutually considered to reward him for his extra exertion.

There is no doubt that the graded payment of the contract and bonus systems conduces to quicker work, and this is most important in the deep levels, where, before the reef can be touched, shafts of considerable depth have to be sunk. With a higher speed in sinking not only is it proved that the actual cost of sinking is reduced, but there is a direct saving of the

interest on the capital of the Company, which only awaits the producing stage to declare dividends. The greatest speed in sinking, whether by hand or machine drilling, has been obtained by the bonus payment.

The amount usually paid each miner participating in the sinking of a shaft is 10s. for each foot above the normal, which may be put down as 80 feet per month; in addition, the miner gets his ordinary day wages.

As sometimes 150 or 160 feet[1] have been sunk per month, it is seen that the amount of the bonus is sometimes considerably more than the daily wage; the natives who may be engaged on these shafts get no monetary reward officially. Payment proportionately to the amount of gold in the ore which has been won—that is, tribute work—is not used on the Rand.

The "time" of the white men is generally kept by a timekeeper, who at some hour during the shift visits each working place; "overtime" is rarely allowed, but when it is, the foreman in charge takes count of the time. Wages are for convenience paid once a month with the salaries, even though a man is working on day wages. In contract or bonus work, measurements are usually made at the end of every week, but the accounts are only made up at the end of the month, when also the complete measurement of the work is made.

Bonus work is further referred to in the index.

Native Labour.—The following schedule of native wages is that which was adopted by a joint committee of the Witwatersrand Chamber of Mines and the Association of Mines of the late South African Republic,[2] on the recommendation of the Association of Mine Managers, and which came into force on 1st June 1897.

[1] During the month of May 1898 the Howard shaft of the Simmer and Jack West was sunk 183 feet.

[2] These two bodies became united under the title of "The Chamber of Mines of the South African Republic."

SCHEDULE OF NATIVE WAGES

Mine

	s.	d.
Machine helpers	1	8
Hammer boys	1	6
Shovel boys	1	3
Tram boys (10-feet trucks)	1	2
Tram boys (16-feet trucks)	1	6
Dry shaft and winze boys	1	8
Wet shaft boys	2	0
Wet shaft boys when developing only	2	6
Boys cutting hitches for timber	1	6
Timber boys	1	2
Stope ganger's assistants	2	0
Station boys where white man employed	1	2
Station boys where no white man employed	2	6
Air-hoist drivers	2	0
Pumpman's assistants	1	8
Platelayer's assistants	1	6
Pipeman's assistant	1	6

Surface

Stokers (12 hours)	2	6
Stokers (8 hours)	1	8
Engine-cleaners	1	6
Sorting boy	2	0
Headgear boys where white man employed	1	4
Headgear boys where no white man employed	2	6

Mill

Elevator boys	2	0
Vanner boys	2	0
Mill boys (12 hours)	2	0
Mill boys (8 hours)	1	4
Banket and sluice boys	2	0
Crusher boys	1	4
Surface trammers	1	9
Mule-drivers	2	6

Cyanide

Solution shed boys	1	4
Boys filling and discharging	1	9
Zinc-cutters	1	6
Tramming residues	1	4

General

	s.	d.
Fitter's boys	1	6
Blacksmith's boys, strikers	2	6
Blacksmith's boys, helpers	1	4
Carpenter's boys	1	2
Mason's boys	1	4
Police	2	6
Compound cooks	2	0
Drill packers	1	0
Drill sorters	1	6
Surface labourers	1	2
Office and store boys	2	6
Assay office boys	2	6
Coal boys, off-loading	1	6

Notes

Timber boys assisting in timbering shafts to be paid at the rate of wet and dry shaft boys.

Seven and one half per cent of the natives employed may be paid special rates.

Month to be reckoned as consisting of at least thirty working days.

As stated above, this schedule of wages came into force on 1st June 1897; most of the costs mentioned in this book were incurred before that date, and when the native wages were such that the schedule given above is a 30 per cent reduction all round, which higher scale of wages was in force from 1st October 1896 to 30th May 1897. Before October 1896 the wages were about 25 per cent higher still.

The sums given are "per shift," of which the hours of actual labour vary. It was proposed in October 1896 to keep the natives underground for nine working hours, but this decision has been relaxed, and the boys now come to surface when their work is done; for hammer-boys this is now considered to consist of drilling a hole 3 feet deep, which can be done by a good hammer-boy in six hours. The shovel and truck boys generally stay underground for about nine hours. On surface the length of the shift is usually ten hours. In

sinking the deep-level shafts the day is usually divided into three shifts of eight hours each, as stated before.

In addition to the actual wage of the native he is also supplied with food at a cost equal to about one-sixth of his wage. It was agreed amongst the mine managers that the allowance of mealie meal per boy per day should be two and a half pounds, and that in addition to the amount which would at that rate be calculated on the number of boys working, 25 per cent extra should be added for boys not working. The allowance of meat is two pounds per boy per week.

Quarters are provided for them in the shape of a large compound, which is generally a wood and iron structure of rooms enclosing a square courtyard in which the cook-houses are placed; such a compound is shown in the photograph which faces this page. In charge of the compound there is a compound manager, an official engaged to look after the natives, who are, as far as order and well-being are concerned, entirely under his control. In addition, the compound manager has to do his best to keep up a good supply of boys, and to distribute those new-comers which he receives to the work for which he has been advised that natives are required. He speaks at least one native language as well as English and generally Dutch, and he is in most cases an Africander. Several able-bodied natives, who, as a rule, in addition to their own native language can also speak one or more of the other dialects, assist him in his work. In one or two cases the late Government had conferred some police powers upon compound managers.

The supply of natives being now hardly equal to the demand, the various companies are put to considerable expense in getting them. The principal sources of supply are from the Portuguese East Coast, from which 50 per cent are stated to come, from Zululand, Basutoland, and Zoutpansberg. Those from the East Coast are considered the best mining boys, because, coming farther, they stay the longest.

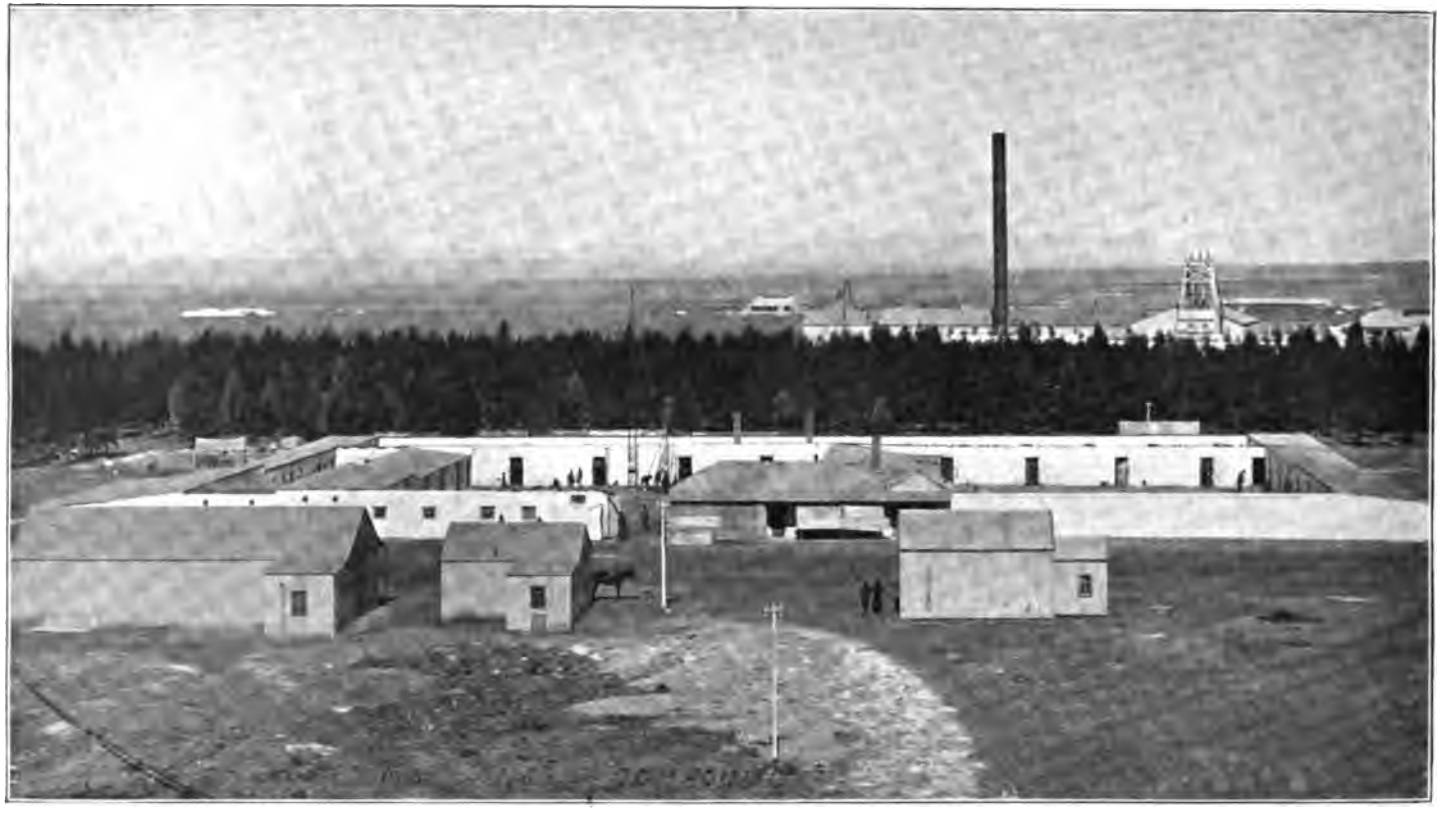


FIG. 198.—KAFFIR COMPOUND OF THE MAY CONSOLIDATED GOLD MINING COMPANY.

It is reckoned that from the East Coast it costs altogether about 62s. 6d. to get a boy to the Rand, this cost being made up of a passport charge of 27s. 6d. to the Portuguese Government, of the railway charges, expenses of food, and the payment of sums for the goodwill of various persons. This high cost promises to be reduced in a short time; the cost of bringing natives from the other places is also comparatively high.

It is usual to charge the boys with all these expenses, but even after they have been received in the compound there is such considerable desertion, that a large portion of the outlay for native labour is never recovered, and in consequence the total cost of native labour is somewhat higher than would follow from the schedule of wages given above.

Stores and Material.—The list of stores and material consumed on the Crown Reef and given on page 433 may be taken to represent the consumption generally of the producing mines on the Rand.

The proportions of the various items for developing mines where a lot of construction is still going on would be somewhat different from those given in that statement.

The prices of labour have already been given; the following is a short statement of the prices of the principal stores which were ruling when the various works whose costs have been given throughout this book were in operation[1]:—

	£	s.	d.	
Coal	1	0	0.00	per ton of 2000 lbs.
Dynamite, 75 per cent nitro-glycerine	4	5	0.00	per case of 50 lbs.
Blasting gelatine	5	7	6.00	" "
Fuse	0	0	5.00	per coil of 25 feet
Cyanide of potassium	0	0	11.00	per lb.
Zinc for cyanide process	0	0	8.50	"
Mining timber, poles	0	0	0.90	per 1 inch diam. and per foot in length

[1] Dynamite has since been reduced by 20s. per case, and coal has also been substantially reduced, whilst other articles have experienced fractional reductions.

	£	s.	d.	
Timber, deals 9″ × 3″	0	0	7.50	per running foot
„ Oregon pine	0	3	3.00	per cubic foot
„ Pitch pine	0	4	0.00	„ „
Drill steel	0	0	5.10	per lb.
Candles	0	11	0.00	per box of 25 lbs.
Iron, common	0	0	2.00	per lb.
Galvanised iron	0	0	6.50	per running foot
Piping, 2 inches	0	0	10.50	per foot
„ 3 „	0	1	4.50	„
„ 4 „	0	1	11.00	„
„ 6 „	0	3	6.00	„

With regard to the cost of machinery, it is found that all engines, etc. imported from Europe are about 100 per cent dearer in the Transvaal Colony than in Europe, owing principally to the heavy railway charges from the coast.

Mr. Seymour, in his evidence before the Commission, stated that the estimated costs for the equipment and development on an average deep-level mine having 100 stamps would be

Construction	£231,000
Development	197,000
Total	£428,000
	= £4280 per stamp

and for 200 stamps

Construction	£315,000
Development	236,000
Total	£551,000
	= £2755 per stamp

These figures do not include the price of the ground. They have been given here because they may be useful for comparison with estimates in other countries.

CHAPTER XXI

DEEP-LEVEL MINING

THE position and prospects of deep-level mining are best studied along the Central Rand from Langlaagte on the west side to near Boksburg on the east.

The payability of the outcrop mines along this stretch is seen from the statement on p. 460, which, with the exception of column No. 1, is taken from the *Standard and Diggers' News*, 30th September 1897, London edition. To the economic status of the average outcrop mine the prospects of deep-level mining must be referred for comparison.

Considering only the reefs at present payable, the gold content per reef claim, whether outcrop or deep level, depends upon the total thickness of workable banket in the claim, upon the average assay value maintained over that thickness—these two together constituting the value of the ore payable body—and upon the dip of the reefs.

I found, from careful computation, first, that the average total stoping width of banket being worked on the Langlaagte Estate, Crown Reef, Ferreira, Henry Nourse, Jumpers, Geldenhuis Estate, New Primrose, and May Consolidated mines was about 94 inches, and that over this width the average assay value was 12.75 dwts., and, secondly, that in the following deep levels along this stretch, the Langlaagte Deep, Crown Deep, Robinson Deep, Nourse Deep, Jumpers Deep, Geldenhuis Deep, Rose Deep, and Glen Deep, these average figures would be respectively 84 inches and 13 dwts.

THE CENTRAL RAND

An Analysis of 25 Outcrop of Mines, 30th September 1897

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15		16	
COMPANIES.	Total Reef Area in Claims.	Intact Outcrop Claims.	Detached Deep-Level Claims.	Estimated Value of Deep-Level Ground per Claim.	Capital.	Price.	Total Market Valuation.	Selling Price per Claim.	Estimated Life in Years.	Tons of Ore crushed.	Claims exhausted.	Yield of Ore per Claim exhausted.	Total Dividends paid.	Dividends paid out per Claim.	Dividends declared for 1897.		Estimated Further Dividends in 1897.	
															Per cent.	Amount.	Per cent.	Amount.
				£	£	£	£	£	Years.	Tons.		Tons.	£	£		£		£
Glencairn	88	80	..	...	500,000	2½	1,250,000	15,600	11	351,000	8	44,000	90,000	11,250	...	...	...	...
May	51	36	...	...	275,000	2⅛	584,375	16,200	9	600,000	15	40,000	110,000	7,333	...	...	...	...
New Primrose	103	78	2	7,500	300,000	4⅛	1,237,500	15,600	11	1,200,000	25	48,000	452,444	18,097	20	60,000	30	90,000
Stanhope	10	2	...	...	34,000	11/16	23,375	11,600	1	146,000	8	18,000	97,750	12,218	...	...	...	...
Geldenhuis Main Reef	11¼	6¼	...	...	150,000	13/16	121,875	12,100	5	153,000	5	31,000	45,000	9,000	10	15,000	5	7,500
Geldenhuis Estate	60	45	...	...	200,000	4 5/16	862,500	19,100	7½	784,000	15	52,000	255,750	17,050	15	30,000	25	50,000
Treasury	26	24	...	...	540,000	3⅛	421,875	17,500	13	90,000	2	45,000	...	...	...	...	5	27,000
Jumpers	25	14	23	10,000	100,000	4⅞	487,500	18,400	4½	716,000	11	60,000	146,500	13,318	25	25,000	15	15,000
New Heriot	49	41	...	...	113,114	8	904,912	22,000	12½	318,000	8	36,000	352,926	44,115	75	83,898	25	27,966
Henry Nourse	28	24	26	7,500	125,000	8⅝	1,078,125	36,700	6	260,000	4	70,000	100,000	25,000	50	62,500	60	75,000
Spes Bona	13	10	...	...	235,000	⅞	205,625	20,500	5	80,000	3	27,000	...	...	...	...	...	...
Geo. Goch	65	55	...	...	325,000	1⅝	528,125	9,600	15	520,000	10	52,000	...	...	...	...	...	...
Wolhuter	169	160	...	...	860,000	5⅞	1,155,625	7,200	30	381,000	9	42,000	13,000	1,444	..	...	7½	63,000
Meyer and Charlton	33	27	...	...	85,000	5⅛	435,615	16,100	7½	374,000	6	62,000	243,808	40,634	20	17,000	20	17,000
City and Suburban	150	135	...	...	1,360,000	5⅝	1,727,500	12,800	17½	778,000	15	52,000	341,000	22,733	5	68,000	5	68,000
Jubilee	20	12	18	10,000	55,000	6¾	371,250	15,900	6	380,000	8	45,000	220,475	27,559	30	15,000	30	15,000
Salisbury	13	7	18	10,000	100,000	2¼	287,500	15,300	4	212,000	6	35,000	80,300	13,363	...	...	...	...
Wemmer	30	21½	23	5,000	80,000	8⅜	670,000	25,800	6	335,000	8½	40,000	200,440	23,581	25	20,000	50	40,000
Ferreira	49	39	...	...	90,000	20½	1,822,500	46,700	12	442,000	10	40,000	726,700	72,670	150	135,000	150	135,000
Worcester	9½	5	..	...	95,722	2⅝	251,270	50,200	4	209,000	4½	46,000	237,126	52,694	20	18,746	10	9,373
Robinson	133	109	...	...	2,750,000	8	4,400,000	40,000	17	880,000	24	40,000	1,970,937	89,588	7	192,500	7	192,500
Bonanza	11¼	10¼	...	...	200,000	4 3/16	837,500	83,700	4	61,000	1	50,000	50,000	50,000	25	50,000	50	100,000
Pioneer	5½	2½	37	4,000	21,000	10	210,000	24,800	2½	148,000	3	49,000	182,700	60,900	350	73,500	125	26,250
Crown Reef	60	40	51	4,000	120,000	12¼	1,470,000	31,600	8	1,052,000	20	40,000	701,900	35,095	170	204,000	...	...
Langlaagte Estate	80	65	...	...	470,000	4⅛	1,938,750	29,800	14	1,480,000	15	98,000	1,128,780	73,252	15	70,500	15	70,500
Totals and averages	1292½	1048½	198	...	9,183,836	...	23,283,297	21,000	9¼	11,950,000	244	49,000	7,747,536	31,700	...	1,140,644	...	1,029,089

With the single exception of the Robinson Deep, these mines just mentioned belong to the first row of deep levels, so that as a whole they may be considered as representative of that group.

The values given are averages obtained by reducing the assay value of the clean banket proportionately to the amount of waste interbedded with it and which comes away with it in mining; they do not relate to the ore to be milled, because the mines sort out a varying percentage of the waste, and where the reefs are mined in separate stopes it is usual to find a somewhat greater proportion of the richer reef being milled, so that a comparison of the ore body in the different mines on the basis of the value of the ore milled is vitiated by the different mining policies; nor do these values include those areas either of Main Reef, Main Reef Leader, or South Reef which are found to be at present unpayable. Along this stretch the Main Reef Leader is being worked in all the mines, the South Reef is worked chiefly on the western, and the Main Reef, while it receives attention all along, is nevertheless worked chiefly on the eastern half.

From these figures it would appear that on an average the deep levels mentioned are working 10 inches less than the outcrop mines, though the assay value is slightly higher, the relative values of the two being as follows:—

Outcrop mines	.	94 in. × 12.75 dwts.	= 1198 in. × dwts.	
Deep-level mines	.	84 „ × 13.00 „	= 1902 „	„

or in round figures as 12 is to 11.

These figures, although they point to a diminution in the value of the payable ore body in depth, are not sufficiently well founded to base upon them a definite statement to that effect; they have been introduced here with the considerations centred around them in order to suggest a good basis for the comparison of different mines, viz. the value of the payable ore body. This value depends upon the absolute value of the ore

body, and, in addition, upon the working costs, because as there are in the Main Reef Series all grades of banket from highly payable to barren, of which those which are unpayable are not mined, it follows that any reduction in the working costs brings about the admission of a further amount of banket as payable (see Appendix I., p. 485). In view of the good returns[1] being made by the deep-level companies, it is fair to consider that the payable ore body in those deep levels is comparable to that of the outcrops.

The gold content per claim, in addition to the value of the ore body, also depends upon the dip of the reefs. The average dip along the surface from the Langlaagte Estate to the May Consolidated is 56°; from this high angle the reefs flatten to an average of 30° in the lowest levels of the outcrop companies, so that the average dip throughout these mines would be the mean of the two angles, 56° and 30°, viz. 43°. The average dip of the reefs in the deep levels mentioned above is 32°. As the gold content varies inversely with the cosine of the angle of dip, that of an outcrop claim along the part of the Rand under consideration must be to that of a deep-level claim (first row) as cos 32° is to cos 43°, or as 118 is to 100.

[1] A comparison of the returns for the first five months of 1898 from representative outcrop and deep level companies along the Central Rand :—

	Value of the Recovery from all Sources, in Shillings.							Value of the Recovery from all Sources, in Shillings.					
	Jan.	Feb.	Mar.	Apr.	May.	Average.		Jan.	Feb.	Mar.	Apr.	May.	Average.
Langlaagte Estate	26.08	33.92	35.71	37.32	33.27	33.26	Langlaagte Deep	..	..	..	..	..	..
Crown Reef	52.06	53.40	52.71	51.26	48.80	51.64	Crown Deep	39.91	36.64	40.31	42.18	37.88	39.38
Ferreira	84.24	88.90	79.70	89.18	84.86	85.37	Robinson Deep	..	..	..	..	63.10	63.10
Henry Nourse	59.19	66.76	64.88	64.62	66.24	64.34	Nourse Deep	52.86	52.08	44.00	44.36	43.95	47.45
Jumpers	33.95	35.44	31.96	34.18	36.23	34.33	Jumpers Deep	..	..	36.91	46.04	47.32	43.42
Geldenhuis Estate	41.74	43.92	46.72	44.22	42.05	43.73	Geldenhuis Deep	39.33	38.96	37.48	38.76	39.52	38.81
New Primrose	28.06	29.26	27.00	30.75	29.90	38.99	Rose Deep	45.67	44.92	42.08	43.06	45.81	44.30
May Consolidated	31.50	37.36	37.44	42.74	44.45	38.70	Glen Deep	..	..	..	..	..	..
				General average		48.79					General average		46.07

The returns of the Langlaagte Deep and Glen Deep, when they are crushing, are not likely to increase the average of the deep levels. The outcrop companies, during these five months, sorted out somewhat more waste than the deep levels.

With regard to the loss of reef by dyke intrusion, and by shattering consequent upon dynamic movement, it may be said that, while receding from the granite which lies to the north of the outcrop, and which was directly or indirectly the cause of the tilting of the beds, the reefs may be the less disturbed, yet there is evidence that in deep-level areas there is extensive longitudinal reverse faulting along dyke intrusions, around which detail faulting and shattering will occur, so that, with regard to the loss of reef due to the presence of dykes and faults, outcrop and deep-level ground would appear to be similarly situated. The upthrows in deep-level ground which are so often found with longitudinal reverse faulting, affect that which constitutes a deep level, *i.e.* the depth of the reef.

Observations of the temperature of the air in deep levels show that the rate of increase in depth (see p. 389) is such that the resultant temperature at 5000 feet would not interfere with comfortable working.

With regard to water underground, the uniform nature and regular bedding of the beds overlying and underlying the Main Reef Series ensures freedom from any great influx of water in these directions, as no water is found collected in them; it has only been along faults and dykes that water has poured in to any extent, but as in all such cases the influx has after a time subsided to a comparatively insignificant amount, it may be stated that deep-level mines as propositions have nothing to fear from such water.

Comparing, then, the features of the occurrence of the payable banket, we see that the ore body in the deep levels has much the same value as in the outcrop mines, but that in consequence of a flattening of the dip the average gold content per deep-level claim is less than that per outcrop claim in those mines under consideration in the proportion of 100 to 118. Further, that in relation to dykes, faults, disturbances, water, and other natural features, the two positions of deep-level and outcrop ground are similarly situated, and that the

mineralogical nature of the banket—neglecting the oxidised zone now practically worked out—and the details of its bedding are in each case identical.

In addition to the gold content per claim the costs of winning the gold have to be considered.

In development, deep-level mining requires, beyond that which is necessary for outcrop mining, the sinking and the equipment of vertical shafts down to the reef.

In actual mining, in breaking the ore and bringing it to the bottom of the vertical shaft, the ore in the deep levels will be won by the same methods and at the same cost as when outcrop ore is brought to the surface, but with deep-level ore the cost of bringing it to the surface will be the additional expense.

In reduction, deep-level ore, being identical with that now being obtained from the outcrops, will require nothing beyond the complete reduction plant of the outcrop companies, and its gold will be extracted by the same methods and at the same cost as with outcrop ore.

These facts may be condensed by saying that in deep-level mining the costs of sinking, of equipping, and of transmitting everything through vertical shafts is all that is above and beyond that necessary for outcrop mining.

Mr. John Hays Hammond has estimated that in the case of a deep-level mine of 250 claims, and with two shafts, each 2500 feet deep, the extra capitalisation per claim which would be required to pay for the sinking and equipment of such shafts would be about £800 per claim.

Such a property would have one shaft for each 125 claims, which for the second row of deep levels, where shafts of such a depth would be necessary, is perhaps a minimum claim area per shaft, so that the sum of £800 would not likely be exceeded. For properties on the third row this extra capitalisation would be greater, though the greater speed which is being obtained in shaft-sinking, and the less cost of shaft-sinking—which two factors have now practically placed the

second row upon the same terms as were existing when the first row of deep levels were started—may in the future place the third row of deeps upon the same terms as now exist for the second row.

Reference to column 14 of the statement on p. 460 shows that the average amount of dividends paid out by the principal outcrop companies per claim exhausted is £31,770. This profit per claim, reduced by the lesser gold content due to the lower dip in the deep levels and by the extra cost of transmitting everything through vertical shafts, is well able to bear this extra capitalisation of £800 per claim.

It is seen on p. 269 that of the cases there given, the hauling and pumping costs ranged from 8.59d. per ton milled to 2s. 3.42d., with an average of 1s. 7d. It may be taken that the average vertical depth was in those cases about 600 feet. The vertical hauling through 2500 feet would at this rate be about 6s. 6d., but with more proper winding and pumping equipments this additional cost to mining would doubtless be reduced to one-half, or 3s. 3d., which amount would also be sufficient to cover all the extra costs due to greater losses in the transmission of compressed air, electricity or steam underground, and to the conveyance of labour, etc.

It must, however, be considered that the working costs now are not what they were whilst on an average the sum of £31,700 was being paid out per claim exhausted; the period over which that dividend extended may be put down as averaging in 1894, since when a saving of about 9s. per ton has been made, so that the extra cost of transmitting everything through the vertical shafts has already been more than doubly set off by the decrease in all-round working expenses, leaving a balance sufficient to relieve the deep levels of any disadvantage under which they lie by reason of their lower angle of dip. It would thus appear that were the second row of deeps set to work in a day at current costs, they would do as well as the outcrop mines have done.

The success which is now attending the exploitation of deep-level ore (first row) augurs well for the second and even the third row; it seems likely, granted the value of the ore body is maintained, and despite the heavier costs of hauling and pumping which will be experienced in the third row of deeps, that when they are in a position to crush they will stand every chance of keeping up the profits per claim.

It has been experienced in several cases that deep-level areas have benefited by upthrows which have brought the reefs nearer to the surface than they would otherwise have been. On the opposite page there is a statement of the principal longitudinal dislocations throughout the Central Rand, from which it will be seen that they are of considerable extent, and that, with but one exception, that of the Metropolitan Fault, they are accompanied by considerable upthrows. These dislocations are indicated on the map facing p. 100. It is quite possible, as mentioned on p. 99, that, in addition to these, the Geldenhuis Estate Fault may change its strike to become a longitudinal fault, in which case, as it is accompanied by reverse faulting, there will be an upthrow of about 600 feet.

In the Bezuidenville bore-hole, put down very near the boundary line of the farm Doornfontein to the south of the Meyer and Charlton, the Main Reef Series was struck at a depth which does not indicate any upthrow along the line between the bore-hole and the outcrop. As will be noticed from the map, none of the longitudinal dislocations would appear likely to cross that line.

In view of figures on the above-mentioned statement, it is apparent that the depth of the reef in deep-level areas depends partly upon the longitudinal reverse faults.

The subject of the formation and the question of the recurrence of these faults are mentioned on pp. 106, 107, and 108.

It is interesting to compare the average dividends paid

Statement of the Principal Longitudinal Dislocations along the Central Rand

Name of Dislocation.	Extent.[1]	Average Upthrow.[2]	
1. Ferreira—Crown Deep Dyke	Ferreira to Langlaagte United, 3 miles	200 feet	In series, resultant upthrow 870 feet.
2. Grahamstown Dyke	Jubilee to Langlaagte United, 4 miles	400 „	
3. South Rand Dyke	Robinson Deep to the Langlaagte Dam, 2 miles	270 „	
4. George Goch Dyke	George Goch	120 „	
5. George Goch Fault	George Goch to Nourse Deep	260 „ downthrow	In series, resultant downthrow 130 feet.
6. Metropolitan Fault	George Goch to Nourse Deep	390 „	
7. Simmer Dyke	Simmer and Jack to Knights, $3\frac{3}{4}$ miles	400 „	In series, resultant upthrow 1000 feet.
8. Knights Longitudinal Dyke	Knights	150 „	
9. East Rand Dyke	New Blue Sky to Simmer East, 5 miles	600 „	
10. Balmoral Longitudinal Dyke	Balmoral	400 „	

[1] No dyke is limited to its extent as here given, which is but the inferred or observed extension.

[2] The average upthrow is given because the upthrow varies; for instance, that dyke numbered 1 has an upthrow of 50 feet in the Ferreira, 100 feet in the Crown Deep, and 450 feet in the Langlaagte United. The throw in no case includes the movement which results from the thickness of the dyke; for instance, in the case of the South Rand Dyke the strata on the south side of the dyke are 600 feet higher than they would have been had they continued uninterruptedly down from the surface, though the actual upthrow, *i.e.* movement, along the wall of the dyke is only 270 feet.

out per claim exhausted with those figures in column 4 of the statement on p. 460, which are estimates of the value of certain deep-level claims, and with the following estimate of the value of other such claims belonging to the Rand Mines, Ltd., which appeared in the *Johannesburg Star* in 1898 :—

Number of Claims.	Situated South of	Distance of Northern Boundary from the Outcrop.	Price per Claim taken.	Total Value.
		Feet.	£	£
174	Nourse Deep	4,000	1,500	261,000
132	George Goch and Wolhuter . .	1,500	2,500	330,000
48	Village Main Reef . . .	2,000	6,000	288,000
4	" " " . . .	7,500	1,500	6,000
13	Robinson	2,750	7,500	97,000
3	Langlaagte Deep . . .	4,500	2,000	6,000
15	United Langlaagte Deep . .	3,000	1,500	22,000
111	Crown Deep and Langlaagte Deep	3,750	5,000	555,000
500			3,130	1,565,000

The price per claim was obtained after a fair discount had been allowed off the selling price of adjoining claims, and after due allowance had been made for working capital requirements.

Though the prices at which deep-level claims can be bought are considerably under the value of the dividends which have been paid out by the outcrop companies per claim exhausted, yet as long as the working of such claims has not been projected or provided for, these prices remain fair ones, because during the indefinite time which lapses till such claims are being worked, the capital which would be laid out in their purchase would lie idle.

It is this consideration which to a great extent determines the number of claims which can be included in one block for working purposes, for with a very large block it is harder to get the necessary capital subscribed, and in any case it is difficult to get the work on a very large block to proceed on

a scale commensurate with the larger size of the block when compared with the rate of operations on an ordinary sized block.

On the other hand, the blocks may not be too small, or the amount which it would be necessary to spend for the sinking and the equipment of two shafts would work out at a higher extra capitalisation per claim than would be warrantable.

The claim areas of the outcrop mines of the Central Rand are given in the statement on p. 460; they average 51.6 claims. The average size of the first row of deep levels immediately to the south of the outcrops is about 220 claims, and that of the second row close upon 500 claims.

In the matter of shafts, great economy has been effected as between outcrop and deep-level mines; thus along the outcrop, from the Langlaagte Estate to the Glencairn, there are about 47 main shafts, besides smaller ones, whereas the shafts of the first row of deep levels number about 25, and those of the second row will number still less.

Estimated costs for the equipment and development of an average deep-level mine (first row) are given on p. 458, and further references to deep-level mining are to be found throughout the index.

APPENDIX I

SECTION XXI

Of the Mining Regulations of the late South African Republic

Responsibility in Connection with the Mines

167. The owner of a mine not residing in this State shall give notice in writing to the Inspector of Mines concerned of the name and address of a person residing in this State who shall be his agent or representative for the purposes of this Law, and who shall have the same responsibility as the owner.

The owner of a mine or his representative shall give the name of the mine, company or syndicate to the Inspector of Mines in writing; he shall also furnish the latter (in writing) with the name and address of the manager, who is responsible under this Law for the working of the mine.

Such notice or notices must be given without delay.

The owner of a mine shall give written notice to the Inspector of Mines of any change of agent, and the owner or his agent shall likewise give notice of any change of manager, or change in the name of the mine, company or syndicate, if possible prior to such change taking place, or at the latest within three days thereafter.

168. Where any working is commenced for the purpose of opening a mine, or the working of a mine has been temporarily discontinued or abandoned, or where the working of a mine is recommenced after any such discontinuance or abandonment, the manager, or failing him the owner of such mining property, shall within fourteen days give written notice thereof to the Inspector of Mines.

169. The working of a mine must be carried on under the control and responsibility of a manager, who must hold a manager's certificate, as set forth in section xxii. par. 7.

A mine in which not more than thirty persons are employed below ground at any one time, shall be exempt from this provision, unless the Inspector of Mines, by notice in writing served on the owner of the mine or his agent, requires that it be under the control of a certificated manager.

A tributor working a mine for his own or joint account, or any person

employed by him, shall in no case be allowed to have the management of such mine unless he himself be a manager holding a certificate of competency.

170. Where in any article of this Law no particular person is named as being directly responsible, the responsibility shall lie with the manager.

171. A mine overseer appointed to assist the manager in the direction and control of the underground works, shall hold a certificate as set forth in section xxiii., and he shall have, in the absence of the manager, the responsibilities of the manager so far as all underground works are concerned, but the appointment of such a person shall not relieve the manager of any personal responsibility.

172. No mine shall be worked without a certificated manager for a longer period than fourteen days.

In case for the time being no certificated manager is obtainable, any other capable person not holding a manager's certificate may, with the approval of the Inspector of Mines, be appointed as manager, but for not longer than three months.

173. In every case, where and so long as a mine is worked without a certificated manager as provided for in the preceding Article, the owner of the mine shall be answerable for all the obligations of the manager under this Law.

174. The manager has to provide for the safety and proper discipline of the men employed above and below ground, and shall appoint such persons as may be necessary to carry out the provisions of this Law, or any part thereof, and unless he proves that he has taken all reasonable measures by publishing and to the best of his ability enforcing these rules, he is guilty of an offence against this Law.

175. When the manager of a mine can prove that the necessary means for carrying out this Law have been refused him by the owner or his agent, the latter will be held responsible for the consequences.

176. The manager, or the mine overseer or the shift-boss, acting under the former's instructions, shall at least once in every shift, whilst the men are at work, visit each working place, and shall see that safety is assured in every respect.

SECTION XXII

Of the Mining Regulations of the late South African Republic

Certificates for Managers

177. Every mine manager must be the holder of a certificate of competency obtained under the provisions of this Law, except as provided for in Article 169, par. 2.

178. For the purpose of examining any candidates for certificates of competency, a commission of examiners shall be appointed from time to time

which shall give regular reports of its proceedings to the State Mining Engineer, who will, on the recommendation of the commission, grant a certificate to the candidate. This commission shall consist of five members, viz. of a chairman, who shall be an Inspector of Mines, and of four others, two of whom shall be selected by the majority of mine managers of the inspection-field concerned, and the other two by the State Mining Engineer. All the members of the commission shall be appointed by the State Mining Engineer and shall hold office at his pleasure. Should any difficulty or disagreement arise with regard to the selection of the examiners by the body of mine managers, the State Mining Engineer will also select these from amongst mine managers.

179. No person shall be allowed as a candidate for examination who has not attained the age of twenty-three years, and has not had practical experience in mines for at least five years. Any person, however, who can satisfy the examiners that he has successfully passed through the curriculum of a recognised mining school or academy, shall be exempt from two of the above five years.

180. Should any difference of opinion arise in any matter connected with the examination of a candidate, it shall be decided by a majority of votes of the examiners.

181. Rules for the conduct of such examination will be made by the State Mining Engineer, who shall have power to alter and revoke such rules as occasion may require. The remuneration of every member of the Commission of Examiners shall be £4 per day, and that of the Secretary 5s. per hour.

182. The State Mining Engineer shall determine the fees to be paid by applicants, which shall not exceed £6 for examination and £2 for a certificate.

183. The Commission of Examiners may, without the usual examination, recommend the State Mining Engineer to grant a certificate of competency to a person who furnishes satisfactory proof of his qualifications, provided that such person has been the manager of a mine for at least three years.

184. The State Mining Engineer will grant a certificate of service to any person holding, on the date of this Law coming into force, the position of manager of a mine in which more than one hundred persons have been regularly employed under ground. Application for a certificate of service must be made to the State Mining Engineer within three months of this Law coming into operation.

185. If at any time representation is made to the State Mining Engineer by an Inspector of Mines that a manager holding a certificate under this Law is by reason of incompetency or gross negligence unfit to discharge his duties, the State Mining Engineer may, if he thinks fit, cause inquiry to be made into the conduct of such a manager, and with respect to such an inquiry the following provisions shall have effect:—

a. The commission of appeal as constituted under this Law, section xxvi., shall be a commission of inquiry.

b. The inquiry shall be public and shall be held at such place as the State Mining Engineer will appoint.

c. The State Mining Engineer shall, before the commencement of the inquiry, furnish to the manager concerned a statement of the case on which the inquiry is instituted.

d. The manager may attend the inquiry in person or be represented by an admitted legal practitioner, and may, if he thinks fit, be sworn and examined as an ordinary witness in the case.

e. The commission shall have power to cancel or suspend the certificate of the manager.

f. The commission shall have, for the purposes of the inquiry, all the powers of a court of summary jurisdiction and all the powers of an Inspector of Mines under this Law.

g. The commission may subpœna all such persons as it thinks fit to call and examine for the purpose of the inquiry.

h. A manager may, in addition to having his certificate suspended or withdrawn, be compelled to pay all or part of the costs of such inquiry as the commission may decide.

SECTION XXIII

Of the Mining Regulations of the late South African Republic

Certificates for Mine Overseers

186. Every mine overseer shall be the holder of a mine manager's or mine overseer's certificate. The latter shall be obtainable in accordance with the following provisions, and will be granted by the State Mining Engineer upon the recommendation of the Commission of Examiners duly constituted as per Article 178.

Mine overseers employed as such on the date of this Law coming into force shall, on application to the Inspector of Mines, receive a certificate of service from the State Mining Engineer.

187. An applicant for a mine overseer's certificate must satisfy the Commission of Examiners that he has had at least four years' practical experience in the working of mines, and has had charge or undercharge of the underground works of a mine for at least one year.

He shall be examined, *viva voce*, in ventilation, timbering and general mining, and must produce evidence of his experience, ability, sobriety and general good conduct.

The examiners may waive the provision about the one year charge of works if applicant satisfies them that he is fully competent to undertake the duties of a mine overseer.

188. The State Mining Engineer shall determine the fees to be paid,

which shall not exceed £3 : 10s. for examination and £1 for a copy of a certificate.

189. The rules provided in Articles 180, 181 and 185 for managers shall also apply to mine overseers.

REGULATIONS FOR THE EXAMINATION OF MINE MANAGERS IN THE LATE S.A.R.

Drawn up on the basis of Article 181 of the Mining Regulations (Law No. 12 of 1896)

A.—General Conditions

1. The Commission for the Examination of Mine Managers consists of one Mine Inspector as Chairman, nominated by the State Mining Engineer, and of four (4) other persons chosen and appointed in terms of Article 178 of the Mining Regulations.

2. In order to constitute an examination, at least the Chairman and two (2) members of the Commission appointed must be present. In the event of votes being equally divided, the Chairman shall have a casting vote.

3. Any matter relating to the holding of an examination which is not provided for in the law or by regulations is—in the event of a difference of opinion, according to Article 180 of the Mining Regulations—to be decided by a majority of votes of the examiners.

4. The examination shall be held in the following subjects:—

Knowledge of law	2 points
Mining practice	3 ,,
Mechanics	2 ,,
Mine surveying	1 point
Geology	1 ,,
Mineralogy	1 ,,
Chemistry	1 ,,
Total	11 points

5. A candidate can, at his option, be examined in electrical practice, which shall have the value of 1 point (paragraph 4) in the subject of mechanics, and is to be specially endorsed on the certificate.

6. In the subjects specified in paragraph 4 one examiner alternately shall examine for every point and decide thereon. One or more points can also be given for written answers on "papers" relating to a branch of the subject concerned. The "papers" to be set by the Examination Commission must be approved of beforehand by the State Mining Engineer, and the answers must—for each point or mark of paragraph 4 that shall be scored therefor—be decided alternately by one of the examiners.

7. The Chairman of the Commission has the option of examining in knowledge of law for two (2) points and in mining practice for one point.

8. Examination on the remaining eight (8) points is allotted amongst the examiners present by mutual agreement.

9. In order to pass the examination the candidate must secure at least six (6) points, whereof are compulsory: 1 point in knowledge of law, 2 points in mining practice, and 1 point in mechanics.

10. Candidates who do not pass can be admitted for re-examination after an interval decided upon by the Commission.

11. Such candidates as can produce to the satisfaction of the State Mining Engineer a certificate of efficiency from an authority or School of Mines may be, in accordance with Article 183 of Law No. 12 of 1896, allowed to be examined in knowledge of law alone.

B.—THE SUBJECTS FOR EXAMINATION

1. *Knowledge of Law.*—Gold Law, Mining Regulations, Boiler Law, Explosives Law, Pass Law, Liquor Law, organisation of the Department of Mines.

2. *Mining Practice.*—Prospecting for metals and minerals, exploration work, sinking and equipment of shafts, mine development, methods of ore-winning, mine-timbering, masonry for shafts and mine-reservoirs for water and gas, etc., propping of the surface (this expression is meant to convey the various means employed to prevent the falling in or "settling" of the overlay in a mine), ventilation and gases, explosives, drainage of water, transmission of power (underground), transport or carriage, methods of sorting ores, coals, and other minerals.

3. *Mechanics.*—Construction of boilers and steam engines and other motors, practical effect and management of same, methods and rules for transmission of power, methods of transport above ground, details of mining machinery and methods of erection. Optional: electrical machinery and its application to mining.

4. *Mine Surveying.*—General acquaintance with the methods of surveying mines, and the drawing of mine plans, and their use and application for the economical and safe working of mines.

5. *Geology.*—Acquaintance with geological formations, and their relation towards mining and commerce.

6. *Mineralogy.*—Methods of the occurrence of mineral deposits, precious metals, base metals, common minerals, ores of the metals ordinarily mined—their commercial, scientific, and industrial uses.

7. *Chemistry*, as applied in the extraction of gold and silver from their ores, especially the methods obtaining in this country: general metallurgy.

(Signed) G. SCHMITZ-DUMONT,
Acting State Mining Engineer.

PRETORIA, *9th March* 1897.

REGULATIONS FOR THE EXAMINATION OF MINE OVERSEERS

1. The Examination Commission, as provided in paragraph 1 of the regulations for the examination of mine managers, is—in accordance with Art. 186 of Law No. 12 of 1896 (Mining Regulations)—entrusted also with the holding of examinations for mine overseers. Paragraphs 2 and 3 of the above-mentioned regulations shall also be of force in this connection.

2. The candidates shall—in accordance with Art. 187 of Law No. 12 of 1896—undergo a *viva voce* (verbal) examination before this Commission in ventilation, timbering, and mining work, with special application of the provisions of the Mining Regulations thereto relating, in which examination the subjects given in paragraph 12 of the said regulations under the heads of mining practice and mechanics shall generally serve as points of departure.

3. The Commission decides by vote as to whether the candidate has passed or not.

(Signed) G. SCHMITZ-DUMONT,
Acting State Mining Engineer.

PRETORIA, 10*th March* 1897.

REGULATIONS FOR THE EXAMINATION OF ENGINE-DRIVERS UPON MINES IN THE LATE S.A.R.

Drawn up on the basis of Art. 107D of the Mining Regulations (Law No. 12 of 1896)

A.—GENERAL CONDITIONS

1. The examinations shall be held at such time and place and in such manner as may be from time to time fixed by the State Mining Engineer (Art. 107D).

2. Every one who wishes to present himself for such examination must make proper application to the State Mining Engineer upon the forms provided for that purpose, which can be obtained at the Mining Inspector's office (Art. 107C).

3. Together with this application must be forwarded (Art. 107B)—

A. A certificate of good character.

B. Proof that the applicant at the date of the examination is not under twenty and not over fifty years of age.

C. A medical certificate that the applicant's senses of sight and hearing are normal, and that he does not suffer from any such mental or physical defect as might have a prejudicial effect on the proper discharge of his duties.

D. Proof that during a year at least he has had charge of an engine or engines used for the ordinary hoisting of minerals.

4. The Commission of Examiners shall consist of five (5) members, nomin-

ated from time to time by the State Mining Engineer. The Chairman of the Commission shall be one of the Government Boiler Inspectors. The four (4) other members shall be:—An engine-driver and a mechanical engineer selected by a majority of the engine-drivers employed on the inspection district in question, a mine manager and a mechanical engineer or an engine-driver appointed by a majority of the mine managers in the inspection district in question (Art. 107E).

5. To hold an examination a minimum of the Chairman and two members of the Commission must be present. In the event of an equality in the voting the Chairman shall have a casting vote.

6. Any matter relating to the holding of an examination which is not provided for in the law or by regulations is—in the event of a difference of opinion, according to Art. 107E of the Mining Regulations—to be decided by a majority of votes of the examiners (Art 107E).

7. The examination is to be as far as possible verbal. The Commission, or one of its members appointed for the purpose, is required to prove by practical trial the efficiency of the candidates in the management of boilers and engines.

8. The Commission is to decide by vote if the candidate has passed or not.

9. As many as four (4) candidates can be examined simultaneously.

10. The examination fee is fixed at two pounds (£2) sterling, payable in advance (Art. 107I).

11. The State Mining Engineer shall issue an engine-driver's certificate of efficiency to each applicant who is duly reported by the Commission of Examiners to have successfully passed the needful examination. A sum of 10s. is to be paid for such certificate (Art. 107F and I).

B.—Subjects for Examination

I. Plain reading and writing.

II. Acquaintance with Arts. 18 to 41 of the Mining Regulations (hoisting work, chapter v.), and with Arts. 96 to 107 (machinery, chapter ix.). Special acquaintance with shaft signals.

III. General acquaintance with the provisions contained in Arts. 14 to 58 of the Boiler Law (Law No. 3 of 1896).

IV. General acquaintance with—

A. Various types of engines—whether single, double, compound, or condensing; especially hoists and hauling engines, direct or indirect acting.

B. Maintenance and care, management, starting, reversing, stoppage by compression of steam through reversal of link motion, handling of winding-engine for changing the skip or cage from one level to another.

C. Causes of refusal to act and of getting out of order in winding-engines. How to act in case of imperfect action of brakes and so forth.

D. Pistons, cylinders and appurtenances, valves, and such like. Their action

as regards the methods of setting an engine in motion (steam reversing gears), clutches, shafts, drums, bearings, wire ropes, and so forth.

V. General knowledge of—

A. Various types of boilers.

B. The care, cleansing, the management and maintenance of fittings (gauges, dash-cocks, etc.), the effecting of minor repairs, methods of firing, and so forth.

C. Feed of water into boilers, various feed arrangements of effecting the same, priming, causes and consequences of the same, blowing down (of mud, etc.), removal of scum (through scum-cocks, etc.), and so forth.

VI. Electrical hoisting and haulage. This is optional, and, in the event of a successful examination, must be specially endorsed on the certificate (Art. 107A).

(Signed) G. SCHMITZ-DUMONT,
Acting State Mining Engineer.

PRETORIA, *20th March* 1897.

ADVANTAGES OF SORTING

EXTRACT FROM THE GENERAL MANAGER'S REPORT FOR THE YEAR ENDING 31ST MARCH 1898, CROWN REEF GOLD MINING COMPANY, SHOWING THE ADVANTAGES WHICH RESULTED FROM SORTING DURING THAT YEAR.

Sorting Plant

A 25-foot circular sorting table was erected last year, and started work on 1st April 1897. It has fully come up to expectations as regards efficiency, but owing to the fact of its being added to an already existing headgear, which necessitates tramming the ore to the table, and raising the ore after it is sorted to the rock-crushers, the costs have not been as low as in some other plants. During the year, 36,364 tons, equal to 16.413 per cent of the rock hoisted, were sorted out. The average assay value of the waste sorted out was practically 1 dwt. per ton. As this plant has played the principal part in earning the increased profits for the past year, it may serve a useful purpose to show how a sorting plant will yield greater profits than can be made by adding more stamps.

The following shows the cardinal facts to be taken into consideration :—

Average assay of residues exclusive of slimes not treated . . .	1.321 dwts. per ton
Tons of waste discarded . . .	36,364 tons
Average assay value of waste	1 dwt.
Total fine gold in waste . . .	1,818.200 ozs.
Total cost of all reduction processes, including rock-crushing and transport	£61,376 3 2
Total cost per ton of all reduction processes, including crushing and transport	0 6 7.546
Total cost of sorting	£5191 10 10

If the total tonnage hoisted had been crushed, 20 additional heavy stamps would have been required. In which case we would have had :—

36,364 tons more residues, at 1.321 dwts. per ton, or 2,401.842 ozs. further gold loss, which equals .		£10,087 14 7
On the other hand the fine gold in the waste, or 1,818.200 ozs., would be gained, which equals .		7,636 8 9
Leaving a net loss on the gold output of 583.642 ozs., or .		£2,451 5 10
The cost would have been increased as follows :—		
36,364 tons would have had to be crushed, reduced and transported at 6s. 7.546d. per ton, which equals	£12,052 10 10	
From this should be deducted the cost of sorting . .	5,191 10 10	
or a net increase in the expenses of		6,861 0 0
showing the total saving over erecting more stamps to be		£9,312 5 10

If no new stamps were added, and the average grade of rock from the mine had been crushed without sorting, the difference in profits could be arrived at as follows :—

The average grade of the rock as it came from the mine can be calculated as under :—

From the 185,179 tons crushed were recovered	119,245.771 ozs.
In the residues, including slimes lost, there were	13,373.034 ,,
The 36,364 tons waste contained . . .	1,818.200 ,,
Therefore the 221,543 tons mined contained .	134.437.005 ozs.
which equals	12.136 dwts. per ton.

If the recovery from this grade rock is assumed to be as good as the average for last year, we would have recovered 89.913 per cent, or 10.912 dwts.; this equals . . . . . £2 5 9.963 per ton

The expenses under all headings except mining would be the same per ton milled as last year. The expenses under mining would be the same per ton milled as per ton mined last year, consequently our total expenses would be as follows:—

Mining . . .	£0	11	11.744
Transport . . .	0	0	3.738
Rock-crushing . .	0	0	4.112
Milling . . .	0	2	11.288
Cyaniding . . .	0	2	2.651
Slimes . . .	0	0	9.756
General charges . .	0	3	11.517
Mine development .	0	0	8.635

Or the expenses would have been reduced to 1 3 3.441 per ton

Leaving a profit per ton of . . . . £1 2 6.522 per ton

Whereas last year, with sorting, we earned . 1 7 11.618 „

Showing a decrease in the profit per ton, without sorting, of . . . . £0 5 5.096 per ton

On the 185,179 tons milled last year this would amount to £50,226 : 14 : 4.

STEEL ROPES

Detailed specification of a rope much used in the deep levels:—

Ordinary or Lang's lay	Lang's
Circumference of rope	$3\frac{1}{8}$ inches
Right or left hand laid	Right hand
Ore turn in	8 inches
Steel or iron wire	Steel
Galvanised or plain	Plain
Number of strands in each rope . . .	Six
Wire or hemp core	Hemp
Number of wires in each strand . . .	Seven
Wire or hemp core	Wire

Weight per fathom	9½ lbs.
Diameter of each wire S.W.G. or 1000th of an inch	11½ S.W.G.
Quality	Best Patent Improved
Tensile strength per square inch . .	80 to 90 tons

Mining Regulations relating to ropes which will be used in raising or lowering persons, Section v., B., Law 12, 1896 :—

The winding ropes shall be of steel, of the best manufacture, free from any defect, and must permanently possess a breaking strain of at least six times the maximum load they are required to carry.

New winding ropes, as also the connecting attachments between the rope and the cage, skip or other conveyance, shall be carefully examined and properly tested as to their working strength by some competent and reliable person authorised thereto by the manager, and shall be used for the ordinary transport of workmen in shafts only after having been proved reliable in the ordinary winding work. The result of the above examination shall be immediately recorded in a book specially kept for that purpose in the office at the mine.

This book shall also contain the following particulars of winding ropes, put on after the date of this law coming into force :—

Name and address of manufacturer.
Dates of manufacture and purchase.
Description and make of rope.
Date on which rope was put on.
Dates of shortening and re-capping.
Dates of tests after shortening.
Breaking strain of wire on re-testing.
Date when rope was taken off.
Length of rope in meters.
Circumference of rope in millimeters.
Number of strands.
Number of wires in each strand.
Diameter of wire in millimeters.
Class of core.
Breaking strain of rope, in kilos[1] or tons.
Safe working load in kilos or tons.
Weight per meter in kilos.

This book shall contain the signature of the person responsible for the required examination.

The winding ropes must be so cleaned at least once a month as to allow of a close examination.

[1] 1 kilo = 2.2 English lbs. avdp.
1 English lb. avdp. = 0.45 kilo.

WORKING COSTS AT THE FERREIRA GOLD MINING COMPANY DURING THE YEAR ENDING 31ST DECEMBER 1897.

	Per ton milled. s.	d.
Mining (including sorting)	19	1.769
Development redemption (125,326 tons milled)	5	0
Transport	0	4.712
Reduction	4	4.278
Cyaniding	2	10.082
	31	8.843

Extract from the Chairman's speech at the annual meeting :—

"In this connection I suppose we shall have to bear the usual onslaught on our working costs. Let me say that we are prepared to meet all comers, and will prove that ours is one of the most economical workings on the fields, where the shareholder is getting handsome returns to-day and is assured of his position in the future. Unfortunately, not alone the public, but many boards of directors and mine managers require education on this head. Gentlemen, in great measure the matter of working costs hangs on the question of sorting. If you make due allowances for local circumstances in each individual mine and have satisfied yourselves that your mining and milling and the various other departments are carried on in the best mode applicable to such individual case, you come back to the great question of sorting, and if you get to the bed-rock on the business you will find that it is all stuff and nonsense comparing working costs in the rough-and-ready manner which is in vogue. Take this mine and let me elucidate what I mean : To commence with the mining of ore—it costs as much to mine the rock discarded in sorting as that passed through the mill, and if we discard one-third of the ore mined, the cost of mining, reckoned on the tonnage milled, appears 50 per cent higher than that of a company mining under similar conditions but not sorting. Then, again, our manager finds that it pays us better, owing to the richness of the ore, to take it all out and timber as we go along, thus adding to the cost of mining as compared with a mine in which pillars are left to support the ground, while at the same time we add to the life of the mine and to the profits to be derived from it. Again, we fill our abandoned stopes with waste to make all secure behind us, and to avoid the renewal of the timber supports, which also adds to the cost of mining as compared to a mine in which pillars are left. The cost of sorting, which amounts for the year to over £5600, is an additional charge to mining which does not occur in a mine where no sorting is done, and is comparatively a small item in mines where only the larger pieces of waste are discarded in sorting. Milling costs per ton milled are higher on account of sorting, because the selected

ore is harder and much more difficult to crush than the discarded waste rock. Cyaniding the tailings is more expensive on account of sorting, because the sand enriched by sorting requires longer and more expensive treatment. Then, again, the cost of concentration by Frue vanners, which amounted to about 10½d. per ton milled during the year, has been proved to be essential on our mine, and is a cost which has not to be borne by companies milling low-grade ore, or where all the sands go direct to the cyanide works. Well, allow us for these, and we have shown you that we are not very far out in our costs, and when you take the question of sorting and our expenses on that head, then you will see where we score. We discarded, during the year, with our sorting appliances, 61,596 tons of rock, assaying 11.88 grains of gold per ton, thus bringing up the grade of our millable rock by nearly 8 dwts., or 46½ per cent, and the effective working capacity of our mill on this increased grade of rock by over 32 per cent. We were thus enabled to crush 125,326 tons of high-grade rock against the same tonnage of usual mine rock, showing a return of £554,764 against £374,633 had there been no sorting. The manager, in his report, shows the working expenses per ton for the past year to be 1s. 3.628d. less than for the previous year; these figures are based on the tonnage and working expenses shown in the report, but there are other circumstances to be taken into account which prove the reduction to be much greater. Prior to 1897 it was the custom on these fields to take a 20 cubic feet truck-load of ore to be equal to a ton of 2000 lbs., but from several tests made we found that an ordinary truck-load was 1900 lbs. dry weight, and this weight was taken by us during last year. Again, the cost of development during 1897 was 2s. 11.03d. per ton, against 4s. 4.1d. in 1896, whilst we have redeemed all ore milled for the two years at 5s. per ton. Taking these facts into account, it shows the actual reduction in working expenses to be 4s. 5.214d. per ton. The slimes plant was started about last Christmas; the completion of this work was considerably delayed through the non-arrival of parts of the machinery, caused by the engineering strike in England. The manager proposes to have the first clean-up in April, so I can only report to-day that the work is going well, and we may look for favourable results. The total milling costs for the year of £27,299 : 9 : 9 include the cost of renovating the old 40 stamps, which have now been converted into the heavier type, thus making the old and new parts of the mill uniform in weight."

A table showing the value of the ore and the percentages of extraction is given on the next page.

TABLE GIVING THE AMOUNTS OF FINE GOLD WON, WITH THE PERCENTAGES OF EXTRACTION

	Total Ozs.	Total Ozs.	Per Ton.		Per Ton.		Per cent.	Per cent.
			dwts.	grs.	dwts.	grs.		
Estimated fine gold in ore milled	...	160,647.73	...		25	15.28	...	100.000
Recovered.								
Fine gold extracted by amalgamation	84,223.83	...	13	10.58		...	52.427	...
Fine gold extracted by concentration (96 per cent of assay value)	23,547.51	...	3	18.187		...	14.657	...
Fine gold extracted by cyaniding	27,532.53	135,303.87	4	9.45	21	14.21	17.138	84.223
Unrecovered.								
Fine gold in slime, cyanide residues, and unaccounted for	...	25,343.86	...		4	1.07	...	15.777

ORE TONNAGE

Tonnages per claim calculated upon one foot of reef (true thickness) and for every 5° of dip from 0° to 85°, allowing 12 cubic feet to the ton.

Dip Degrees.	Reef Area per Claim.	Tons (2000 lbs.) per Claim.
	Square Feet.	
0	64,025	5,335
5	64,270	5,356
10	65,017	5,418
15	66,288	5,524
20	68,131	5,678
25	70,642	5,887
30	73,927	6,160
35	78,157	6,513
40	83,580	6,965
45	90,553	7,546
50	99,602	8,300
55	111,626	9,302
60	128,051	10,671
65	151,495	12,625
70	187,195	15,600
75	247,378	20,615
80	368,703	30,725
85	734,602	61,217

THE REDUCTION OF WORKING COSTS

There is some banket of the Main Reef Series, for instance the great bulk of the Main Reef west of the Henry Nourse and that of the South Reef east of the New Heriot, which, lying quite close to other banket which is being mined, is yet not sufficiently auriferous to be worked at a profit. As there are all grades of banket, from that which is highly profitable to that which is barren, it follows that any reduction in the working costs is followed by the admission of a further amount of banket as being payable.

The table given on p. 443 shows the reduction which has been effected in late years, and which, up to the year 1896, was chiefly due to progress in the perfection of methods and machinery. This progress will undoubtedly continue, but it is considered evident that the consequent reduction to present working costs will be but fractional, and especially so when compared with that which would result from reductions in the prices of labour and stores, and further, it is considered that this fractional reduction will be balanced by the increase to present costs which will be caused by the extra hoisting and pumping, etc., when the deep levels are the principal producers.

This position is shown by the comparison of the working costs of 1896 and 1897. It is seen from the above-mentioned statement that between those two years the total reduction was two shillings per ton milled. This was almost entirely due to the smaller amounts expended on native labour and coal, as is seen from the following statement, which was taken from the speech of the chairman of the Rand Mines, Limited, 24th March 1898:—

STATEMENT OF COSTS PER TON MILLED

	1896.	1897.
White labour	8s. 3.00d.	8s. 4.30d.
Native labour	6s. 9.50d.	5s. 4.00d.
Explosives	2s. 11.50d.	2s. 11.50d.
Coal	2s. 4.50d.	1s. 11.25d.

This smaller expenditure was due to the considerable reductions which were made in the prices of these two commodities during the year 1897, so that the progress in the perfection of methods and machinery made during that year was responsible for a very small reduction on working costs.

APPENDIX II

THE BANKET OF THE TARKWA GOLDFIELD, WEST AFRICA

THE reefs which have been worked in the Tarkwa, Adjah Bippo, and other mines of the Tarkwa Goldfield, have the same structure and occurrence as the reefs of the Witwatersrand Goldfields, and they have doubtless been formed in the same manner, so that, though there are differences in exact composition, the reefs of the Tarkwa Goldfield can rightly and most fittingly be described as banket reefs.

This term has now become associated with reefs of even value, regular thickness, and assured extent. Though these characteristics are possessed by the banket of the Tarkwa Goldfield, it is more than probable that the Witwatersrand Goldfields will always remain unique by reason of their vast aggregate value.

POSITION AND EXTENT

This goldfield is situated in the Gold Coast Colony, about forty miles inland by road from Axim or Sekondi, to which latter port there is railway communication. At present this railway is only opened as far as the station of Tarkwa, which is situated in the south-western portion of the goldfield, but it is also practically complete in its extent along the line of reef, beyond which it passes on to Kumassi.

The banket formation commences at the Tamsoo mine, from which it passes in a direction about N. 40° E. through Effuenta, Tarkwa, Abbontiakoon, Abosso, Adjah Bippo, Cinnamon Bippo, and Chida, to Busumchi, beyond which it has not yet been traced. The distance from Tamsoo at one end, to Busumchi at the other end, is about twenty miles, and along this stretch the outcrop of the reef can be followed by pits which have been made by the natives in their search for ore. At several places there are interruptions to the continuity of these pits, but these are not of insurmountable extent, nor are they great enough to make the identity of the reef across them doubtful. This line of reef may be considered to be the extent of the goldfield. There are other occurrences of conglomerate which are doubtless extensions of this line, but they have generally been found to be too poor to be economically interesting; they stand in the same relation to the line of the

Tarkwa Reef as the reefs at Venterskroon do to the reefs in the neighbourhood of Johannesburg; they are on the other side of the syncline, and they present more interest geologically than economically.

The line of reef, which may be called the line of the Tarkwa Reef, dips towards the north-west, as indicated by the arrows upon the sketch map which faces this page.

The variations in the angle of dip at the various mines are as follow:—

Chida	40°-60°	at the	outcrop.
Cinnamon Bippo	50°	,,	,,
Adjah Bippo	60°	,,	,,
Abosso	45°	,,	,,
Abbontiakoon	34°	,,	,,
Tarkwa	28°-35°	,,	,,
Effuenta	34°	,,	,,
Tamsoo	20°	,,	,,

In addition to these variations along the formation, the dip also varies across the formation and in a most regular manner: in crossing the formation in the direction of the dip, the angle of dip decreases. From the evidence of bore-holes, it also appears that the angle of dip decreases in depth. These two facts suggest at once the existence of a syncline of which this line of reef is the south-eastern edge. Thus, at Adjah Bippo the dip of an underlying bastard reef exposed in the railway cutting, is vertical; the dip of the Tarkwa Reef is 60°; and the dip gradually decreases in this direction, till it becomes quite horizontal, and then turns up to incline in the opposite direction.

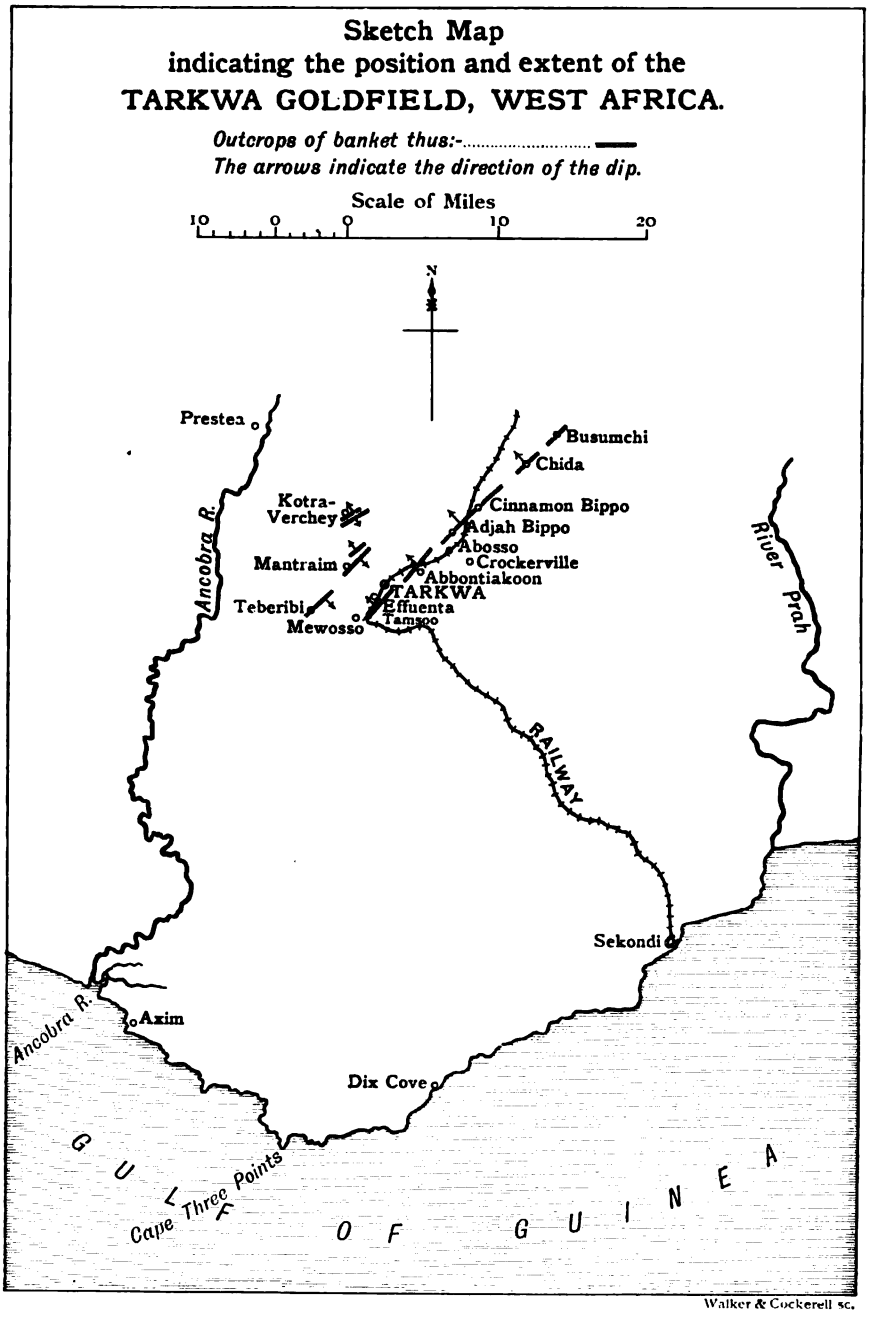
Along the same line of section, the dips decreased in Bore-hole D from an average of 56° in the upper portion of the bore-hole to one of 46° where the reef was cut at 1216 feet. These features are shown in the section which faces page 490, Fig. 199.

At Abbontiakoon also, the dip has flattened from 34° at the outcrop to 28° at the bottom of their deepest bore-hole.

In the south-western portion of the goldfield the syncline is complete; the reef which dips into the ground between Tamsoo and Tarkwa reappears again on the other side between Teberibie and Mantraim, with a dip in the opposite direction. At Teberibie the reefs dip 35° to the south-east, and at Mantraim they dip at a lower angle in the same direction.

A few miles farther to the north in the same district, there is upon the Ajapang hills an anticline of the banket formation, one portion of which dips south-east towards Mantraim to form a second syncline, and the other portion of which dips towards the north as the beginning of what may be a third syncline. These outcrops are indicated on the sketch map which faces this page.

Along the line of section through the Adjah Bippo mine, there seems to be no doubt that where the Huni River is crossed, about three and a half miles from the outcrop of the Tarkwa Reef, the dip on the surface is in a



Walker & Cockerell sc.

north-west direction; but it is stated that at an intermediate point there are dips in the opposite direction—to the south-east, and that at a distance of about 15,000 feet back from the outcrop there is the axis of an anticlinal bend or break, which anticline, however, was not sufficiently pronounced to bring the reef to surface. Such an occurrence, though not yet confirmed, appears very reasonable, as the axis of such an anticline would be in good line with that which is more surely indicated at Mantraim. It is represented in the section which faces this page, Fig. 199.

Farther to the north-east, no outcrop of the other side of the main syncline has yet been discovered, and in view of the presence in that neighbourhood of a large intrusive mass, it is quite possible that the banket formation, instead of being bent, has been sheared by the intrusion and has not been brought to surface. At Cleary's Camp, which lies about four miles to the north-west of Busumchi, there is a pronounced outcrop of the sandstone formation, dipping to the north-west. The relation of this outcrop to the main line of formation is not yet clear.

To the south-east of Chida, which lies near one end of the reef, and to the south-east of Tamsoo, near the other end, there are occurrences of banket which make it possible that new synclines may be found on this side of the line of reef; and this is rendered more likely because at the former place the sandstone formation has been found dipping to the south-east.

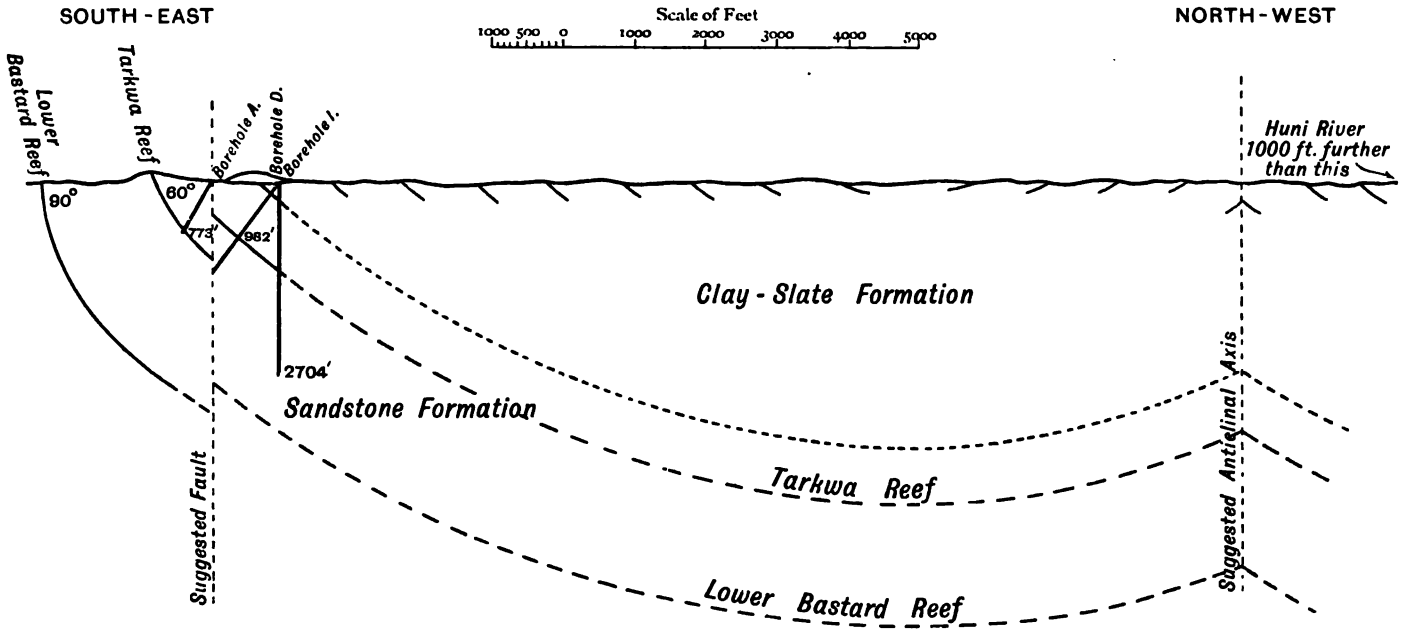
Geological Structure and Formation

The banket beds are found in a sandstone formation which lies upon an igneous basement rock and under a clay-slate formation. The sandstone and the clay-slate are of sedimentary origin.

The basement rock on which these sedimentaries lie and against which they incline, consists chiefly of basic quartz-diorite and porphyrite, these rocks occurring in all conditions from perfect preservation to extreme alteration. They can be seen on the railway from Sekondi to Tarkwa, from the fifth mile to the thirty-sixth mile, in which extent several quartz veins have been exposed in the railway cuttings. Similar rocks have been noticed in many places around the Tarkwa Goldfield, and there is no doubt that they form the basement rock over a very great area in this country. Many of the quartz veins are auriferous, and of such, the two most interesting occurrences in the neighbourhood are at Prestea, which lies about fifteen miles to the north-west of Tarkwa, and at Crockerville, which is not far to the south-east of Adjah Bippo. At the former place a good deal of development work has been done and a good deal of quartz has been opened up, some of which contains visible and coarse gold. The reef there has a well-maintained extent in a north-east and south-west direction, parallel with the general trend of the banket formation. It is an interesting fact that most of the important reefs in the country do thus conform to the same general strike.

Fig. 199.

SECTION ACROSS THE FORMATION AT THE ADJAH BIPPO MINE, SHEWING THE WAY IN WHICH THE BEDS MAY OCCUR.



With regard to the reef at Crockerville, not so much is known; it has, however, produced some pieces of white quartz containing much coarse and visible gold.

The sandstone formation is of a peculiar character. In appearance at the surface it is usually grey in colour and gritty in texture, and it is at once evident that, in addition to quartz, there is a large felspathic content which renders the rock much softer than sandstones usually are; in the neighbourhood of the reef, however, it is generally harder. Under the microscope, quartz and felspar are seen to be the principal constituents, in addition to which there are many iron grains, magnetite or ilmenite, and enough calcium carbonate to make most pieces of this rock effervesce slightly upon treatment with acid. The formation has generally the appearance of an altered sedimentary, of which the angular fragments and the amount of decomposing felspar suggest that the detritus has not travelled far. The average specific gravity is 2.75, which is higher than that of ordinary sandstones, this being doubtless due to the amount of contained magnetite or ilmenite.

In this sandstone formation there are occasionally many thin white quartz veins, but such veins have not yet been noticed to occur in the overlying clay-slate. Generally speaking, they do not contain gold, but it is quite possible that in places they have been associated with gold, because sometimes many old native pits have been noticed to have been sunk where there was evidence of many of these quartz veins.

It was noticed that nearer to the clay-slate the sandstone was finer grained, though the line of demarcation between the two formations was always quite distinct. It may, however, here be mentioned that in the clay-slate itself there were occasional bands which were more sandy than clayey.

The clay-slate is a phyllite; it contains plates of mica, some chlorite, a good deal of magnetite or ilmenite, many brown grains which are probably of leucoxene, and a little quartz, all these minerals being set in a felspathic ground-mass.

Though these sedimentaries have been thus simply divided into a sandstone and a clay-slate formation, many occurrences of rock were met with in bore-holes, of which the origin and the class were most difficult to determine.

In bore-hole D, to the south of the Adjah Bippo mine, clay-slate was first encountered, then a greenish rock which effervesced strongly with acid, then a large thickness of sandstone, then the greenish rock again, then sandstone enclosing the reef, then a thickness of a crystalline quartz-diorite, then 40 feet of the greenish rock, and lastly a schistose metamorphic rock in which the hole was stopped about 1500 feet below the reef.

In bore-hole I, which was put down from the same point on the surface as D, but at an angle of 53° from the horizontal, D being vertical, a very similar series was met with, so that a satisfactory correlation was made between these two bore-holes, from the surface down to a position where in bore-hole I the reef was struck, whereas, judging from the results of bore-hole

D, there should have been a considerable thickness of the greenish rock over the reef. With bore-hole A, in the same line of section, no satisfactory correlation could be made. The greenish rock has in thin sections under the microscope the appearance of an epidiorite. The following is the sequence of rocks in bore-hole A :—

26 feet of clay and sand.
40 ,, boulders and sand.
72 ,, fine-grained sandstone.
22 ,, calcareous felspathic rock, probably epidiorite.
11 ,, fine-grained sandstone.
225 ,, coarse-grained sandstone.
71 ,, felspathic rock, probably epidiorite.
306′ 6″ of coarse sandstone.
3′ 7″ of Banket (773′ 6″ to 771′ 1″).
312′ 4″ of coarse sandstone.

Small pebbled banket occurred at 282, 640, 641, 647, 648 (8″ of banket), 652, 665, and 668 feet.

In the bore-holes on the Effuenta and Abbontiakoon properties similar rocks were met with, and, in addition, there were some layers of a true pinkish dolomite with which quartz veins were sometimes noticed. The epidiorite was a greenish rock of igneous appearance, and in one of the cores from Abbontiakoon a speck of gold was visible in it. The beds of epidiorite and dolomite appeared to occur both above and below the reef, and in the clay-slate as well as in the sandstone; as, however, owing to the position of the reef, the bore-holes have been more in the sandstone than in the clay-slate, more is known about the occurrences in the former than about those in the latter formation.

The most interesting points about No. 3 bore-hole, Effuenta, were as follows :—

0-134 feet—clay, sand, and decomposed sandstone, then hard sandstone.
237-262 ,, epidiorite with white quartz veins.
355-382 ,, four brecciated reefs, with hard sandstone.
434-443 ,, epidiorite.
468-492 ,, banket zone, in which there were several layers of banket, between which, towards the top, the partings were chiefly of coarse talcose sandstone, and, towards the bottom, of quartz seams and quartzite.
704-739 ,, three bands of epidiorite.
1033 ,, six inches of quartz.
1051 ,, bottom, in sandstone.

In all of the bore-holes the change from one rock to another was usually abrupt, and so distinct that the angle of dip could often be taken from the line of contact.

A. R. Sawyer, in a paper on "The Tarkwa Goldfield" read before the

Institution of Mining Engineers on May 30, 1902, gives the following table of the formations occurring in this goldfield in stratigraphical order :—

Clay-slate Formation—
- Sandstone, fine-grained, green and white, not thick.
- Clay-slate, over 1000 feet thick.
- Sandstone, fine-grained, green and white, very thin.

Banket Formation—
- Consisting principally of sandstone, more or less coarse ; quartzite ; conglomerate more or less auriferous ; slate, not more than a few inches thick ; and some calcareous beds.

The two formations, sandstone and clay-slate, are very noticeable on the surface ; they form ridges which are the most noticeable features in the contour of this goldfield. It has been mentioned before, that in the neighbourhood of the reef the sandstone is harder than usual, and so it has resulted that that portion of the formation stands out well as a strong ridge all along the line of reef. This ridge falls steeply down the north-west slope under the reef, and more gently down the south-east slope overlying the reef, till in this latter direction another noticeable ridge rises parallel to the first, and this ridge is of clay-slate.

The different distances from the outcrop of the reef to the commencement of the clay-slate formation are as follows : —

At Cinnamon Bippo,	with a dip of	50°,	about	900 feet.
At Adjah Bippo,	,,	60°,	,,	1400 ,,
At Abbontiakoon,	,,	34°,	,,	1000 ,,
At Effuenta,	,,	34°,	,,	1000 ,,

At these two latter places there appears to be a second clay-slate ridge set still farther back, and at Cinnamon Bippo and farther to the north-east there is also a second ridge, but this apparently is of intrusive quartz-diorite. At Adjah Bippo, however, which occupies a central position along the line of reef, there is no such second ridge ; from the section which is given through that mine, it is seen that the thickness of the clay-slate formation will be there about 3500 feet ; and this is only the portion which remains, for, with no protecting formation on top, some portion has undoubtedly been denuded away.

With regard to the thickness of the sandstone formation, there would appear to be about 800 feet of sandstone above the Tarkwa Reef, and a further 1600 feet down to the lower bastard reef ; about a mile farther in this direction is the Crockerville mine, where a quartz reef occurs in the basement rock, so that the bottom limit of the sandstone will occur before that mine is reached. It may then be said, that along this section the sandstone formation has a thickness which is more than 2500 feet and less than 7500 feet, and will probably be about 5000 feet.

Displacement, Dykes, and Faults

It would appear that at both ends of the line of reef there has been much displacement. To the south-west, below Tamsoo, the sandstone is micaceous and fissile, as though it had been subjected to much pressure and strain, such as would follow upon dynamic movement. In this neighbourhood there is at Detchikroom a piece of banket reef quite out of line and dipping to the south, which is evidence of great displacement. At the north-eastern end of the line of reef the sandstone has again been rendered micaceous and fissile, and the continuation of the reef has not been found in proper line.

Between these two disturbances, which determine the present known extent of the line of reef, there are many displacements of lesser extent, of which the most noticeable occurs when following the reef in a south-westerly direction from Adjah Bippo. In doing this, the reef on Horton's Abosso does not occur in proper line, but along a line about 2000 feet back to the north-west. Not only is the continuity of the reef thus broken, but the continuity of clay-slate ridge also, this being especially noticed when tracing this ridge up from the south-west.

There is evidence from the depths at which the reef was struck in the bore-holes to the dip of the Adjah Bippo mine, that a longitudinal fault, such as indicated in the section, will probably be found to cross between those bore-holes, along which fault the extension of reef in depth will have been upthrown. Allowing for some flattening of the angle of dip, it was estimated that the reef would be struck in bore-hole D, at a depth of from 1500 feet to 1800 feet, whereas it was struck at 1216 feet. This is perhaps best explained by an upthrow as suggested, which at the same time would account for the apparently great distance between the outcrop of the clay-slate and that of the reef at this place. It could also, however, be explained by a much greater flattening of the dip than was expected.

The greenish rock, of which bands were encountered in many of the bore-holes, has been classed as an epidiorite, a diorite formed by metamorphism of a basic rock. The correlation of these bands between the two bore-holes D and I would suggest that they are bedded, but the one which was found just over the reef in D was not found in I.

Close to the bottom of D there was an occurrence of a basic quartz-diorite which was granularly crystalline and had a specific gravity of 2.99. Below this rock, and adjoining it, there was a considerable extent of the epidiorite. The large outcrop of igneous rock which forms a ridge behind Cinnamon Bippo is also of basic quartz-diorite very similar to that just mentioned.

Details of the Banket

The most important of the banket beds is that known as the Tarkwa Reef. This occurs in the upper portion of the banket formation, being

separated from the clay-slate by a thickness of sandstone which varies from 600 to 800 feet.

Overlying this reef there are many small pebbled banket beds which do not contain gold. Thus in bore-hole A, Adjah Bippo, several of such bands, occupying with the sandstone partings 28 feet of the bore-hole, were struck about 100 feet above the reef. In bore-hole No. 1, Abbontiakoon, bands of brecciated felspar, talc, and quartz, occupying with the sandstone partings 24 feet of the bore-hole, were struck about 100 feet above the reef; and in bore-hole No. 3, Effuenta, four bands of brecciated felspar, talc, and quartz, occupying with the sandstone partings 27 feet of the bore-hole, were also met with about 100 feet above the reef. In bore-holes Nos. 1 and 2, Effuenta, this overlying series is closer down upon the Tarkwa Reef, so that the average true thickness of the sandstone between the two occurrences of banket along the whole line of reef may be put down as about 65 feet.

In addition there are other but smaller occurrences of similar banket situated farther up, but so far no other banket reef has been sectioned by a bore-hole below the reef. In bore-hole No. 2, Effuenta, where there was the appearance of such an occurrence, it is most probable that the lower reef was a duplicated recurrence of the upper reef. In the section, however, which is obtained on surface in the railway cutting underlying the Tarkwa Reef at the Adjah Bippo mine, there is a lower bastard reef exposed, which, as shown in the section, is separated from the Tarkwa Reef by a bed of sandstone 1600 feet thick.

These two occurrences of bastard banket may be described as the Upper Bastard Reefs and the Lower Bastard Reefs respectively.

The Upper Bastard Reefs are characterised by consisting of small angular fragments of felspar, talc, and quartz, these fragments having an average size of about half an inch, and being set in a fine-grained matrix of similar material, with the addition of magnetite or ilmenite. These reefs have at their best only shown traces of gold by assay.

The Lower Bastard Reef is about 18 inches thick, and its pebbles are large and well rounded. They are in greater part of vari-coloured rock, blue and pink, and in lesser part of white and black quartz. The matrix is very felspathic, not containing much magnetite, or ilmenite, and the pebbles are rather sparsely distributed in it. This reef has been sampled, but no gold was found in the samples, which is what might have been expected from the look of it. A similar outcrop, though of a much larger bed, was noticed farther along the railway towards Tarkwa, of which the pebbles were large, and almost entirely of blue and pink rock.

The Tarkwa Reef

In the Adjah Bippo mine this reef consists of a piece of banket from 10 to 25 inches in thickness, which assays on an average about 25

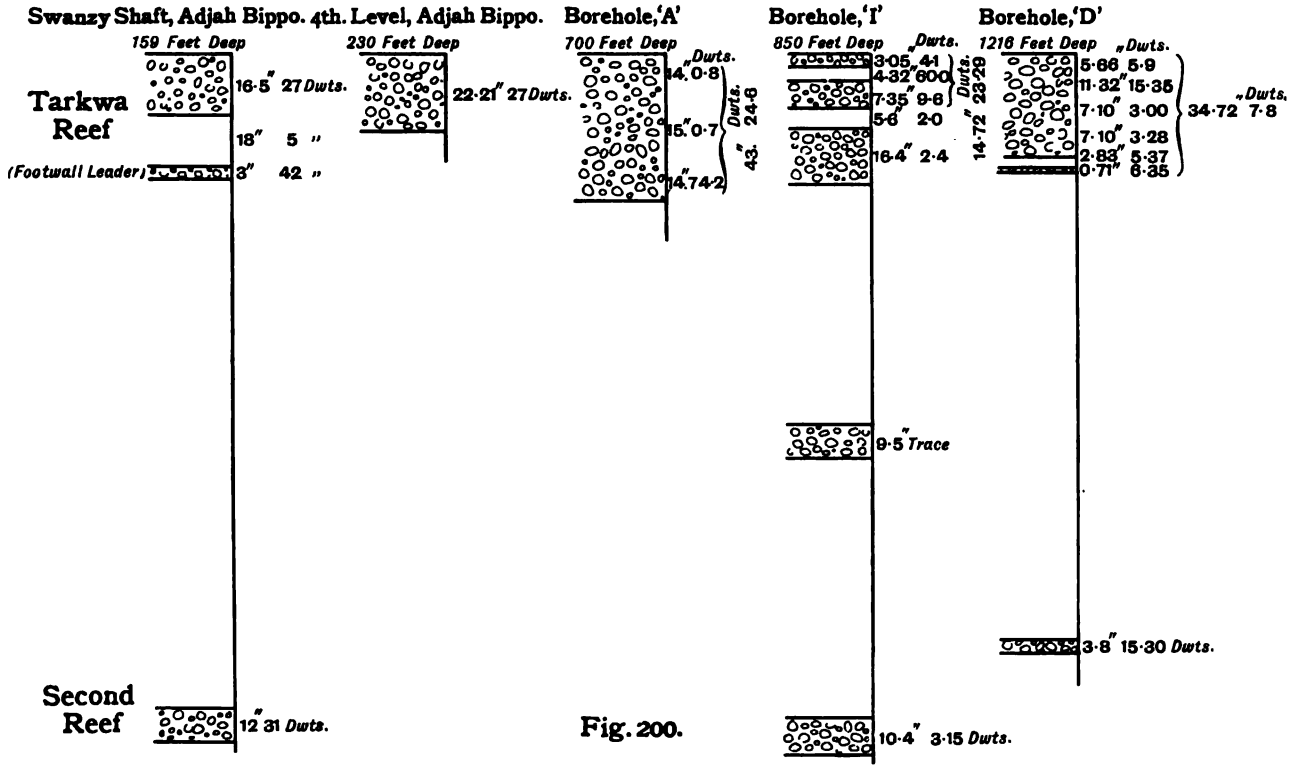


Fig. 200.

Sections of the Tarkwa Reef in the Adjah Bippo mine and in Boreholes to the dip of that mine.

Scale of Feet

Inches 12 6 0 1 2 3 4 5 6 7 8 9 10

The depths given are the computed vertical depths.

dwts. Underneath this there is sometimes a footwall leader, which has a thickness varying from a row of pebbles up to 12 inches, and an assay value which is usually high. Between these two pieces of banket there is from 12 to 18 inches of sandstone, which is striped with layers of oxide of iron, and which assays from 3 to 10 dwts. whether the footwall leader be present or not; and above the reef there is, between it and a very pronounced hanging wall, about 8 inches of sandstone, which assays about 3 dwts.

At places in the mine there is a second reef in the footwall, of which the best occurrence is on the third level, to the north-east of the incline shaft. The horizontal distance between it and the main reef is about 16 feet near the shaft, though this distance lessens farther to the north-east. Below the third level the main reef dips about 57 degrees, whereas the second reef is almost vertical. This second reef is uncertain in extent. It pinches out about 70 feet to the south-east of the incline shaft, and it does not extend far below the third level. In the upper levels it has, however, provided a good deal of ore, as stopes have been carried upon it right up to the surface, where there are two rows of old native workings. Apparently it had much the same size, value, and appearance as the main portion of the principal reef.

These features of the Tarkwa Reef are shown in the sections which face page 496, Fig. 200.

The banket is composed of pebbles which have an average length of a little over one inch, though sometimes they occur up to as much as four or five inches; they are chiefly of opaque white quartz, but sometimes of clear white quartz, black quartz, and more rarely of a dull, opaque, tinted felspathic rock. The matrix is a fine-grained mixture of granular quartz and white mica, with a little decomposing felspar and much magnetite or ilmenite. In addition there is some secondary silica, and, more rarely, some fine needles of rutile. It has been mentioned before, that in the neighbourhood of the reef the sandstone appeared more strongly consolidated than elsewhere in its extent, this being probably due to the impregnation of secondary silica, of the presence of which there is further evidence in some thin quartz veins which generally follow with the formation, but are seen at times to cut across the reef.

On the fourth level, along the drive to the north-east, the reef in the face was very hard and compact, and strikingly similar in general appearance to the reefs on the Rand.

With reference to the present workings of this mine, the reef has proved to be better in one portion than in another, that which extends north-east being considerably better than that which extends to the south-west. This difference in value is probably sufficiently marked to allow of the occurrence of the richer portion being popularly described as that of a richer chute. It is indicated by the following results of a sampling made by Mr. Louis Webb of the fourth level :—

North chute . .	560 feet sampled	23.13 inches	40.44 dwts.
South chute . .	560 „	21.18 „	10.84 „
Average . .	1120 feet sampled	22.21 inches	27.00 dwts.

The gold which has been recovered by amalgamation from this mine has been generally very fine; according to the account of the sales, it has sometimes fetched as much as £4 : 1s. per ounce, at which price its fineness would be a little over 950. The published results of former crushings from this mine record that 50,285 tons of ore have been crushed, and that they have yielded 44,477.58 ozs. of bullion having a value of £177,819. From these figures it is obtained that the average value of the bullion was £4 per oz., which is equivalent to an average fineness of 940. It is also obtained that the value of the yield was 70.72 shillings per ton crushed, and this was by amalgamation alone.

The gold in the ore cannot be described as coarse, nor is it all fine. Upon examination, it can often be seen as specks arranged with grains of magnetite around the pebbles in the ore; but these specks are not easily discovered—they are not ostentatious. Gold does not often occur in the pebbles, and no published statement has yet been made that it has been seen to occur in them; but the association of gold and quartz is so intimate and universal that it would be safe to say that gold does occur in the quartz pebbles. It is not such gold, however, that makes the banket valuable; it is the gold in the matrix which does that.

The following publication by the Abbontiakoon Company bears upon this point:—

"The core of the reef, passed through in No. 4 bore-hole at a depth of 337 feet, has been divided into lengths of three inches each, and assayed with the following results:—

1. 9 dwts. 12 grs. of gold per ton.
2. Nil.
3. 10 dwts. 17 grs. of gold per ton.
4. 8 dwts. 17 grs. of gold per ton.
5. Nil.
6. Nil.
7. 2 dwts. 12 grs. of gold per ton.
8. 6 dwts. 0 grs. of gold per ton.

The sections carrying no gold were solid quartz pebbles."

The description of the character of the reef in the neighbourhood of the Adjah Bippo mine is made complete by the following account of the reefs which were struck in bore-holes A, D, and I:—

In bore-hole A, which was put down at a distance of 892 feet from the outcrop, the reef was struck at an inclined depth of 773 feet. It assayed 27 dwts. over a corrected thickness of 43 inches, though practically all of

this value was shown to be contained in the footwall leader, which assayed 4 ozs. 3 dwts. over a thickness of 14 inches. In this bore-hole there was no sign of a second reef in the footwall.

In bore-hole D, which was put down at a distance of 1748 feet from the outcrop, the reef was struck at a depth of 1216 feet, though it was anticipated from the angle of dip that it would be struck between 1500 feet and 1800 feet. Its average assay, including a small parting of sandstone, was 7.8 dwts. over a corrected thickness of 34.72 inches, and there was no portion of it which was conspicuously better than the others. About 16 feet below this reef the second reef was struck, having a corrected thickness of 3.8 inches and an assay of 15.3 dwts.

In bore-hole I, which was put down from the same position as D, but at an angle of 53° from the horizontal, the reef consisted of three bands of banket separated by partings of sandstone. The two lower bands showed a very low value; the upper band was still poor but better, whereas the best value was found in the parting under the upper band, and this assayed about 3 ozs. This parting made, with the upper layer of banket, a reef having a corrected thickness of 14.72 inches, and assaying 23.29 dwts. In this bore-hole the second reef was struck 16 feet below the main reef; it had a corrected thickness of 10.4 inches, and it assayed 3.15 dwts. The sections of the reefs in these bore-holes are represented on the plate which faces page 496, Fig. 200.

The Cinnamon Bippo property has a lateral extent of 8000 feet along the reef, and it lies about two miles to the north-east of the Adjah Bippo mine. The outcrop of the reef is well seen; it occurs upon the slope of a hill, along which its course is marked by a continuity of pits made by the natives. These pits were started on the back of the reef, and sunk vertically till the reef was struck, after which they were turned on to the reef, which in this mine has an average dip of 50°. In addition to these workings there are adits, which were put in at the bottom of the slope, and from which a good deal of ore has been won. The reef which was exposed in the old stopes and workings varied in thickness between 2 and 3 feet, though it was not usual that all of this thickness was compact banket. The pebbles gave the impression that they were larger than, and not quite so well rounded as those of the Main Reef Series in the Transvaal, in further comparison with which they perhaps were not so evenly laid, giving as a consequence a somewhat larger proportion of cement.

At the outcrop the reef had a general reddish colour, though there were stripes of black iron grains in the walls, and to a less noticeable extent in the reef itself. These stripes were generally parallel with the bedding, but often they showed false bedding, such as is indicated in the following sketch.

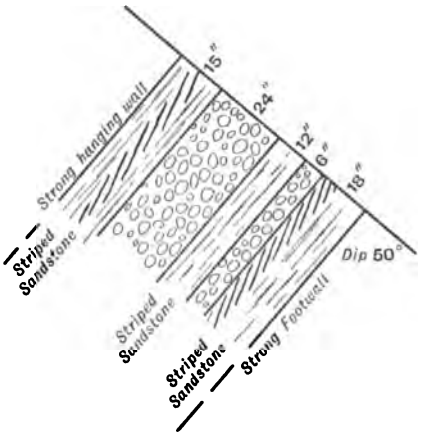


FIG. 201.—Section of the Tarkwa Reef at the Cinnamon Bippo mine.

Some large pebbles in the footwall constituted the footwall leader, which was sometimes well separated from the main portion by as much as 18 inches of sandstone, and at other times was close up under the main portion. Though these pits had been open some time, the two walls were usually in very good repair, and they were very well marked.

In the lower stopes, which had been worked from the adit levels, the reef was very soft, so that the projecting pebbles could easily be knocked out of it. In this place it looked as if the reef had suffered from some hydrothermal action, for some of the pebbles, which were doubtless originally of hard felspar, were now found soft and white. More coloured pebbles were noticed in this mine than at Adjah Bippo, and these appeared to be felspathic rather than siliceous. Two or three quartz veins were noticed to cross the reef.

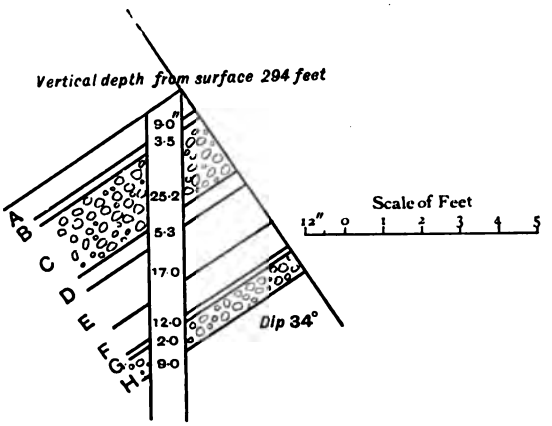
Between Cinnamon Bippo and Adjah Bippo the reef is uncertain; near to the latter mine it is represented by a small and rich leader, but near to Cinnamon Bippo two or three adits which have intercepted the reef have shown it to be pinched and poor.

The value of the reef on this property has been tested by the sampling of such ore as was left in the old workings and by bore-holes.

The average result of sampling the ore at 241 different places, irregularly situated over a total length of 2766 feet, was 9.13 dwts. over a width of 27.15 inches, and the bore-hole results were as follow:—

	B	C	F	H	
Distance from outcrop .	552′	420′	630′	852′	
Angles from the horizontal .	vertical	60°	vertical	43°	
Incline depth of reef . .	662′	234′	1020′	647′,	650′
Computed vertical depth .	662′	207′	1020′	450′,	452′
Width of reef . .	8″	18″	20″	5″,	11″
Assay of reef . .	22 dwts.	24 dwts.	2 dwts.	4.3 dwts.,	20.8 dwts.

In the Abosso mine the average value of the reef on the first level was about 32 inches and 14 dwts., and on the second level 34 inches and 35 dwts. It was generally composed of two or three layers of banket



A, sandstone with iron sand; B, sandstone, considerable iron sand, and a few pebbles, shows gold by assay; C, banket, pebbles up to two inches in length; D, reef matter, core much broken, shows gold by assay; E, sandstone, small pebbles, some iron sand, shows gold by assay; F, soft talcose sandstone; G, white quartz, shows gold by assay; H, banket, pebbles one inch in length.

FIG. 202.—Section of the Tarkwa Reef cut in No. 2 bore-hole, Abbontiakoon.

separated by partings of sandstone. According to the records of this mine, 30,185 tons of ore have been crushed, and have yielded 11,536.8 ozs. of fine gold by amalgamation alone, equivalent to 7.63 dwts. of fine gold per

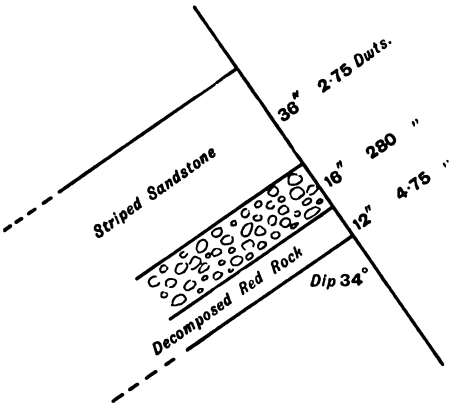


FIG. 203.—Section of the Tarkwa Reef in the Taquah mine; second level, 50 feet south-west of No. 1 shaft.

ton crushed. The fineness of the bullion was somewhat less than that of the Adjah Bippo mine.

The character of the reef in the Abbontiakoon mine is shown by the section, given in Fig. 202, of the reef which was cut in No. 2 bore-hole at 294 feet. Some details of the core from the No. 4 hole have already been given.

In the Tarkwa mine, belonging to the Taquah and Abosso Gold Mining Company, the reef in the No. 1 shaft consists of from 3 to 5 feet of striped sandstone against a well-defined hanging wall; underneath this sandstone there is the footwall leader, which varies from 6 inches to 18 inches in width; beneath this again there is in some parts of the mine a red decomposed rock, and in other parts a soft yellow sandstone. These features are well represented in the section, given in Fig. 203, of the reef on the second level, 50 feet south-west of No. 1 shaft.

A chute of richer ore is seen by the following figures to occur along that level:—

	Thickness.	Assay.		
165 feet N.E. of shaft, footwall leader	6 inches.	0 ozs.	9 dwts.	8 grs.
145 " " " "	6 "	0 "	6 "	10 "
100 " " " "	6 "	0 "	2 "	10 "
54 " " " "	22 "	1 "	18 "	0 "
35 " S.W. " "	12 "	14 "	12 "	0 "
50 " " " "	16 "	14 "	0 "	0 "
180 " " " "	9 "	13 "	0 "	0 "
190 " " " "	12 "	5 "	0 "	0 "

In the neighbourhood of No. 2 shaft the reef is divided into several bands and has quite a different appearance, as is shown by the following section, which was taken on the second level, 100 feet to the north-east of the shaft:—

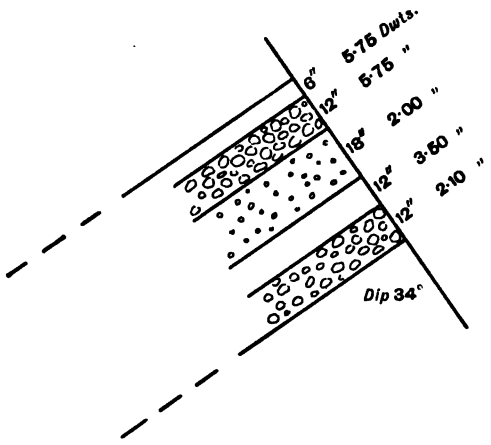


FIG. 204.—Section of the Tarkwa Reef in the Taquah mine; second level, 100 ft. north-east of No. 2 shaft.

In the French tunnel there was the appearance of a second reef of good value lying about 15 feet below the main reef; but this was not certain, as the ground was disturbed.

Between the years 1892 and 1898, 28,065 tons of ore were crushed from this mine, and 20,460 ozs. of bullion were obtained by amalgamation. This bullion contained 19,647 ozs. of fine gold, so that its average fineness was 960. The yield by amalgamation was thus 14.0 dwts. of fine gold per ton crushed.

The reef at Effuenta is exemplified by the core from the No. 1 bore-hole at that mine, which gave the following section :—

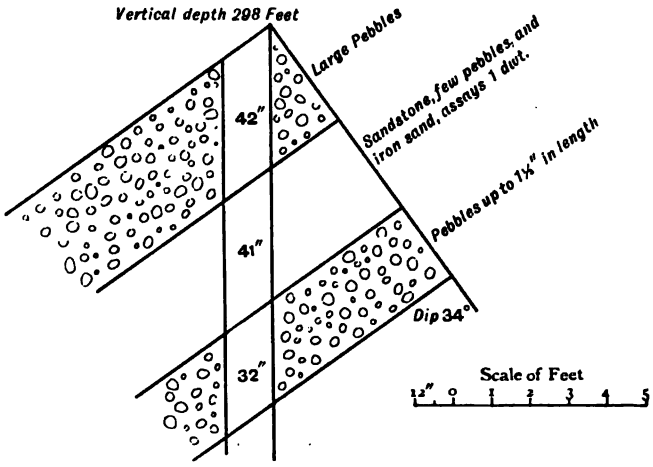


FIG. 205.—Section of the Tarkwa Reef cut in No. 1 bore-hole, Effuenta.

It would appear that, coming from Adjah Bippo towards the south-western portion of the syncline, the reef gradually becomes of greater width, and two bands of banket develop, of which the upper is the larger. At Tamsoo there is one bed of conglomerate about 8 feet thick, in addition to other smaller ones.

The conglomerate beds on the other side of the syncline which exists in the south-western portion of this goldfield are much thicker than those which have been described. At the Teberibie mine two reefs occur: one about 50 feet thick, including much sandstone, and the other about 5 feet thick. At Mantraim the reef is also a large one. The work which has been done upon the Ajapang hills shows the banket to exist there in two beds, of which the lower one is about 4 feet thick, with small pebbles, and the upper one about 19 feet thick, including the sandstone partings. The pebbles of this second reef are larger, being sometimes as much as 4 inches in length.

These large reefs are poor; sometimes good values are found in them, but there is no definite portion of their thickness which remains consistently of higher grade than the remainder—the better values are continually changing from one layer to another.

The Formation at Sekondi

Many pebbles are to be noticed in the town of Sekondi; these come from beds of conglomerate which are exposed on the seashore just under the town. They occur in a sandstone formation which dips about 15° to the north-west and strikes in a north-easterly direction. This sandstone has a purplish or earthy red tint; it does not bear any sign of great age; it is well consolidated and undisturbed, but not compact enough for building-stone. It is seen to be faulted most regularly in places, but no quartz veins were noticed in it.

The bed of conglomerate which is more particularly noticeable is about 7 feet thick, and it can be followed for about 300 yards till it enters the sea. It is composed of different bands, some of which contain small pebbles which are not so well rounded, whilst others contain larger and well-rounded pebbles. The pebbles are generally of semi-opaque white quartz with only a few clear white and a few black ones.

In addition to this one, there are other conglomerate beds overlying, which, judging from the number of pebbles they have shed, must be very much larger than the one described.

In the railway cutting near Sekondi these beds are cut through and a very large extent of them is seen. They have not yet been demonstrated, by samples taken from them, to be auriferous; but it is quite possible that to a very small extent they do contain gold, and they may be the source of the gold which can be obtained from the sand along the shore at this place, which sand contains in addition a good deal of black iron sand.

On the general line of argument that it would be surprising to find two conglomerate formations so similar in character, so similarly situated, and comparatively speaking so close to one another, but belonging to two different geological periods, it is probable that these conglomerate beds at Sekondi will be rightly correlated with those at Tarkwa.

Geological Age

There is so far no positive evidence of the age of the two sedimentary formations with the enclosed banket beds. They lie directly upon igneous rocks of Archean age, and no younger formation has yet been described above them. From the similarity of the banket to that of the Witwatersrand Goldfields, it would be reasonable to suppose, in the absence of anything to the contrary, that they are of the same age, and the consensus of opinion about the age of the Witwatersrand banket is that it belongs to the Devonian period. In support of this idea it may be said that the outcrop of the sandstone beds along the seashore at Sekondi has much the appearance of the exposures of Old Red Sandstone on the Devonshire shores, though these latter make a stronger outcrop.

A. R. Sawyer[1] states his opinion that the banket of the Tarkwa goldfield is of Devonian age, but that the sandstone and conglomerate formation at Sekondi is more recent and probably of Jurassic age.

Origin of the Banket

It seems more than likely that the extent of the banket formation in this part of Africa was once very considerable, and that the portion which now remains would be small if compared with what was once the original whole.

In addition to the occurrences at Tarkwa and at Sekondi, there is a wide distribution of pebbles upon the surface both in other parts of the Gold Coast Colony and in the neighbouring French Colony, the Ivory Coast. These pebbles were in all probability once contained in banket beds which are not existent now.

The most satisfactory solution of the formation of such beds is that of a strong tidal action acting over a great area of flat ground upon debris from igneous rocks containing auriferous quartz reefs and being themselves auriferous in a small degree. The narrow marginal size would be taken from them, and extent would be given to them, by an advance of the sea such as would result from a gradual and contemporaneous subsidence of the land.

Genesis of the Gold

Considering this goldfield by itself, the following facts bear on this question :—

1. That the heavy iron sand, which is undoubtedly a detrital product, is found equally in all portions of the reef, and not towards the bottom only, as might reasonably be argued. It follows from this that detrital gold might also be found in all portions of the reef.

2. That the gold is not large or gross, but occurs in very small grains, specks, and colours, amongst the magnetite grains.

3. That the average fineness of the gold is high, and more like that of detrital gold than reef gold.

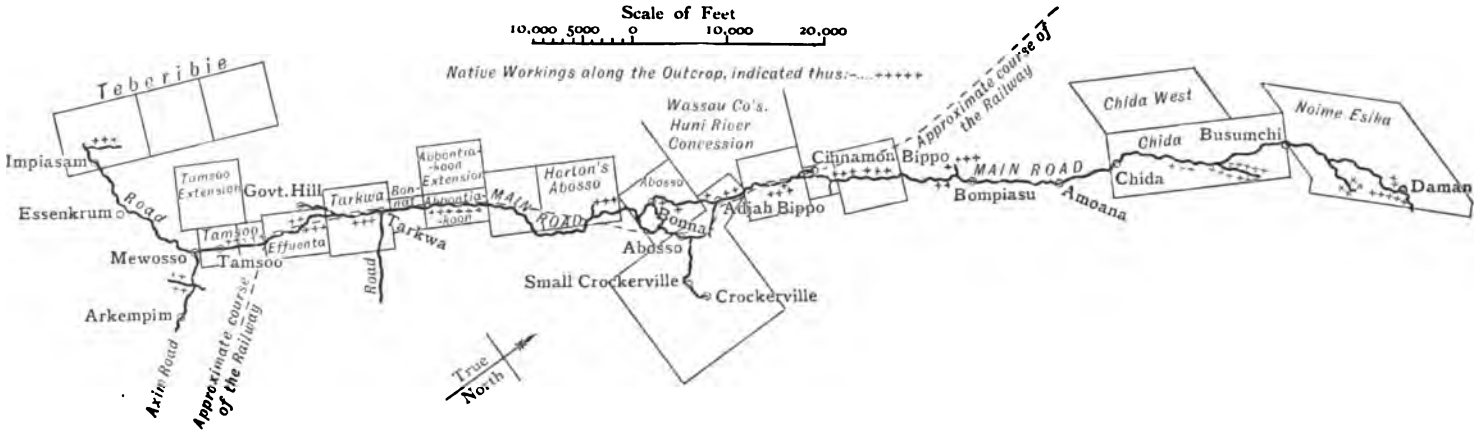
4. That generally the tenor of the banket is very even, and where there is a marked difference in value, the richer ore occurs rather in patches than in chutes. A case was noticed in the Abosso mine where the ore improved in depth, and in the Adjah Bippo and Tarkwa mines, where the improvement was in a lateral direction.

These considerations, when taken together with the fact that the gold occurs in a detrital bed, would lead to the conclusion that the amount of gold in the banket which has made it valuable, is of detrital origin.

With reference to the occurrence in the Transvaal it is, as stated in

[1] "The Tarquah Goldfield, Gold Coast, West Africa," paper read before the Institute of Mining Engineers, 3rd September 1901, by A. R. Sawyer.

MAP SHEWING THE PRINCIPAL PROPERTIES ALONG THE LINE OF THE TARKWA REEF.



Chapter VI., the balance of opinion that the presence of the gold could be most satisfactorily explained by the impregnation theory. This same theory could be applied to the Tarkwa banket, but not so satisfactorily as the detrital theory. By means of it, it would be difficult to explain the deposition of so much gold and so little else—there are practically no mixed sulphides and but very little secondary silica; to explain the general poverty of the thin quartz veins found in the neighbourhood of the reef; to explain the granular character of the gold and its high fineness.

Indeed, it may be that, with the additional data afforded by the Tarkwa Goldfield, the detrital theory may now be more satisfactorily applied to the Transvaal banket. The reefs on the Witwatersrand, though often richest in their lowest portions, contain gold throughout; but the occurrence of the detrital iron sand throughout the thickness of the Tarkwa Reef, shows that such a distribution of detrital gold is quite possible, and that it cannot be taken as an argument against the detrital theory. The banket reefs of the Transvaal contain an amount of gold of undoubted detrital origin,[1] which is proved by its waterworn appearance; and the fact has not been sufficiently appreciated, that much of the pyrites also has the same waterworn appearance,[2] from which it may be inferred that a good deal of the pyrites which now appears too comminuted to have possibly preserved any rounded shape, may be the last result of pieces which have become entirely broken up. The lower fineness of the Transvaal gold can be explained by the separate occurrence of a baser bullion with the pyrites.

In addition to these considerations there are others which become evident only upon comparison of the occurrences in the two goldfields. Thus, banket reefs are conglomerate beds; they differ from the ordinary conglomerate beds in the extent to which they are auriferous and in the extent to which they are composed of quartz pebbles. This would suggest that the entry of the quartz pebbles and the entry of the gold are associated, and the only way in which they could make an associated entry would be as detrital products. The lower bastard reef of the Tarkwa Goldfield contains few quartz pebbles, and, consequently, but little gold.

[1] Page 128. [2] Pages 42, 48, 59, 62, 71.

INDEX

Accidents, 446
Ada May Reef, 59
Adit levels, 287
Administration of mines, 429
Africander Reef, 36, 59
Africander or Keely Series, 61, 62
Agnes Munro, trial crushing, 330
Air compression, 366-373
  compressors, 366
    cross compound, 370
    duplex, 366
    efficiency, 369, 385
    H.P. per drill, 368, 369, 384, 385
    leakage, 368, 384
    pressure of air delivered, 372
  cylinder, 370, 372
  mains, 373
  receivers, 373
  valves, 370, 371
Alexandra Estate Reef, 22, 42
American Rock Drill Company's drill, 133
Amygdaloidal diabase, the, 25
  extent, 25-27, 35
  nature, 28, 65
  stratigraphy, 26-28
Angelo, mill samples, 331
  stopes, 342
Angelo Deep, sinking, 162, 178, 193, 196
Angle bob, 279
Assay plans, 313
  value, terms of, 3
Auriferous igneous rock, 112
  quartz, 43, 74, 127
  quartzite, 51, 68, 69, 309, 408
Axles and wheels, 250, 397

"Backs" of a level, 286
Balance bob, 279
Balmoral Longitudinal Dyke, 105
Banket, 5
  bastard banket, 93
  bituminous matter in, 57-59
  cement or matrix, 42, 47, 49
  gold occurrence, 42, 54, 62, 66, 67, 74, 91, 92, 123-128
  minerals, less important, 42, 48, 65, 67, 73, 74
  nature in Bird Reef Series, 46, 47
    ,, Black Reef, 65-67
    ,, Chimes Series, 51
    ,, Elsburg Series, 49
Banket, nature in Kimberley Series, 48
    ,, Livingstone Series, 45
    ,, Main Reef Series, 68-72, 93
    ,, Nigel Reef, 54
    ,, Quartzite-Shale group, 22
  occurrence outside, 36
    ,, Quartzite-Shale group, 21
    ,, Witwatersrand Goldfields, 41
  ore shoots, 42, 65, 67, 125
  pebbles, 46, 48, 55, 59, 68, 70, 71, 73, 74, 92, 127, 128
  pyrites, 42, 48, 59, 62, 67, 71
  specific gravity and weight, 3, 66, 71
  theories of origin of gold, 123, 126
Banket Beds, 5, 41
  Black Reef, 64-67
  Quartzite-Shale group, 41
  Witwatersrand Beds, 43
    Central district, 43
    Klerksdrop district, 57
    Krugersdorp district, 60
    Nigel-Heidelberg district, 53
    Southern Heidelberg Syncline, 63
    Van Ryn-Chimes district, 50
    Venterskroon district, 56
Banket (mine), reefs of, 90
Bantjes Consolidated, assay book, 314
  shafts, 151, 155
Bastard banket and reefs, 92
  South Reefs, 45, 72, 79
Battery Reef, Nigel district, 54, 63, 64
  Series, 61, 62
Bearers, 173, 175, 281
Becker, Dr., 125, 126
Bedded dislocations (see also Slate Parting), 78, 94, 108-113
Benoni Gold Mines, reefs of, 85
Bezuidenville bore-hole, 139, 145, 148, 466
Big Pebble Reef, 59
Bins (see Ore Bins)
Bird Reef Series, 46, 47, 52, 53, 55, 56, 60-62
Bituminous matter, 57-59
Black Reef, 64-67
Black Reef formation, 29.
  component beds, 29, 32
  extent, 29-31
  inclination, 29-31, 64, 66, 67
  stratigraphy, 30, 31, 65
Bladray electric drill, 380
Blake crusher, 426
Blakneys furnace, 382

Blasting certificate, 431
gelatine, 362
Blocking, in timbering, 196
Bonanza, shaft, 154
stopes, 345
Bonus payment, 157, 291, 292, 363, 452
Book-keeping, 432
Bore-holes, 138, 139
deviation, 141
temperatures, 142
surveying, 143
Boring returns, 146
Boschrand Reef, 29
Botha's Series, 61, 90
Boundary pillars, 349
Box-holes in stopes, 337, 347
Breast stopes, 336, 342
Bucket (pump) rods, 282
Buckets, kibbles, 162, 244, 245
Buffelsdoorn Reef, 57, 60
Bullock diamond drills, 133, 134
Bunkell, H. B., 56
Burness drill-sharpener, 382

Cages, 257, 267
Candle-holders, 392
Candles, 391
Cap pieces, timbering, 178
Carrier belts, 407
Catches for pump rods, 281
Catlin, R. M., 286
Central District, Witwatersrand Syncline, 6, 10
dip of beds, 10, 11, Fig. 29
reefs, 43
Central Nigel Deep, reefs, 55
Central Reef, Nigel district, 55, 63, 64
Certificates, managers', 471
overseers', 473
Chalmers, J. A. (see Hatch and Chalmers)
Chamber of Mines, 443, 453
Chambers, Sydney A., 162
Chimes Mines bore-hole, 53, 105, 139
Chimes Series, 46, 50-53
Chimes West headgear practice, 220, 226, 227, 237
City and Suburban, contracting, 193, 361
crushing and cost, 428
headgear practice, 220, 226, 227, 237
Main Incline shaft, 150, 155, 193, 211
ore-bin doors, 215
pumps, 275
shaft timbering, 178, 180-183
sorting and cost, 424
Claim, the (quartz reef), 1
relation to areas, 2
tonnages of reef, 484
Claim areas of properties, 152, 469
Claims, base metal and others, 447
licences, 447
prices, 468
profits per claim, 465
Climax drills, 377, 378
Coal, cost per ton ore milled, 485
price per ton, 457
Coal measures, 34
Collar set, 173, 183
Combined stopes and stoping, 336, 342
Commonage Reef, 59
Compound manager, 456
Compounds, Kaffir, 456
Compressed air, 367
efficiency with drills, 384
friction in mains, 374
leakage in mains, 368, 375
use in ventilation, 387
Conglomerate bed, 33
Consolidated Main Reef, ore-dressing plant, 421
shafts, 150, 151, 155
Construction work, costs, 448
Consulting engineers, 430, 431
Contact reefs, 21, 54
Contract payment, 192, 193, 291, 292, 360-362, 452
Core books, 149
Core boxes, 145
Cornish pumps, 275
Correlation, of Venterskroon Reefs, 56
aid to, by bedded dykes, 60, 78, 110, 113
of strata in South Africa, 37, 38
Cresswell, F. H. P., 157, 159, 194
Cross-cuts, cross-cutting, 148, 294, 295
Crosse, Andrew F., 143
Crown Deep, bore-hole, 138
ore values, 329, 334
reefs of, 140
Crown Reef, accounts, 432
contracting in stopes, 360
development, cost of, 285, 296
extraction, 333
hoisting costs, 269, 270
Main Reef, 69
ore bins, 212
ore value, 329
recovery value, 411
sorting, cost of, 412, 424, 478
sorting table, 417, 478
stope plans, 365
transport, cost of, 402
wages, 450
winzing, 293
Crusher station, 404, 406
Crushers, 425
Crushing, cost of, 425, 428, 436

De Launay, 44, 49, 123, 124
Deep-level mining, 459
Deep levels, capitalisation due to shaft-sinking, 464
claim areas, 152, 469
costs, development, 458
costs, equipment, 458
costs, hoisting, pumping, 465
mine water, 271
output of gold, 444
shafts, 151-153, 237
value, ore-milled, 462
value, reefs, 332, 459
Deep levels and outcrops, 2
values, ore bodies, 461
values, ore milled, 461
working costs, 464, 465
Denny, G. A., 28, 58, 113
Department of Mines, 445
Depreciation, 303, 304, 432, 442

Detachable hook, 162, 246
Developed ore, amount, 301
definition, 300
Development, 285
cost of, 296, 302, 304, 438, 483
cost in a deep level, 458
excess, 304, 440
methods of charging, 302
range of the term, 285
redemption, 302, 438, 483
Diamond drill, construction, 132
Diamond drilling, 131
methods of recording results, 144
prospecting by, 144
value of results, 140, 141
Diamonds in banket, 58
Dip, effect on gold content, 462
flattening in depth, 12, 14, 16, 106
from drill cores, 15
nature, 11
secondary steepening, 13-15, 106
Dip faults, 94
Dips, across the formations, 12, 23
around Witwatersrand Syncline, 10, 15, 16, Fig. 29
in the deep levels, 12, 14
in Southern Heidelberg Syncline, 18
of Black Reef, 29, 30, 31, 64, 66, 67
of Quartzite Shale group, 20, 23
Dislocation, 94
features, varieties, 94-96
loss of reef by, 302, 463
Dislocations (see also Bedded, Longitudinal and Transverse Dislocations)
delineation on plans, 318
features in Central Rand, 115
principal in Central Rand, 118-120
,, Van Ryn district, 121
,, Lancaster, etc., 122
Dividers, 165, 168
Dogs, gates or keps, 258
Dolomite Formation, the, 32
extent, 35
stratigraphy, 30, 32
Downthrows, 106
Draper, David, 38, 39
Driefontein Consolidated headgear, 220, 221
Drill cores, 139, 140
sharpening, cost of, 189, 361, 381, 382
steel, 359, 381
Drives, distances apart, 286
gradient, 287
intermediate, 287
stope, 287, 341, 347
Driving, machine drilling, 291
contracting, 292
drilling the face, 288
rate, 291
Du Preez Reef Series, 21, 23, 42
Dunn, E. J., 39
Duplex compressors, 366
Duplication, 47, 81, 101-105
Durban-Roodepoort, driving, 291
Incline shaft and timbering, 151, 172, 178, 179, 182, 183
ore bins, 212
pump, 275
reefs of, 89, 90
Durban-Roodepoort, stopes and stoping, 348, 354, 356
Deep, shaft-sinking, 154, 157, 193-195
Dyke, definition of, 94
Dykes, influence on the gold content, 58, 111, 112, 126, 127
petrological character, 100, 102, 103, 108, 109, 110, 112, 113, 115
Dykes and faults, age of, 103, 104, 113, 115
Balmoral Longitudinal Dyke, 105
Crown Deep, Ferreira Dyke, 101, 116
delineation on plans, 317
dip of, 100, 106, 115, 120
East Rand Longitudinal Dyke, 101
Geldenhuis Estate Fault and Dyke, 99
George Goch Dyke, 102
George Goch Fault, 98
Grahamstown Dyke, 99, 101
in the Lancaster, 122
in the Van Ryn district, 121
Knights Longitudinal Dyke, 104
loss of reef by, 302, 463
Metropolitan Fault, 103
Meyer and Charlton Dyke, 100
natural classifications, 120
Robinson Dyke, 100
Salisbury Dyke, 113, 116
Simmer Dyke, 103
South Rand Dyke, 102
tabulated statement, 118, 119
thicknesses, 115
Village Main Reef Fault, 105
Dynamite, 362

East Rand Longitudinal Dyke, 104
East Rand Mines, averaging samples, 322
development, 304
sample book, 314
Elandsheuvel Reef, 60
Elandslaagte Reef, 36
Elsburg Series, 44, 49, 50, 52, 56, 57, 60, 63
Endless chain haulage, 400
rope haulage, 398
End plates, timbering, 165, 166
Engine-drivers' examination, 476
Equipment, deep-level, cost of, 458
Examinations, managers', 474
overseers', 476
engine-drivers', 476
Excess development, 304, 440
Explosives, 362
consumption of, 164, 165, 194, 195, 290, 291, 300, 361, 362
cost per ton ore milled, 485
prices, 457
Extraction, 331
percentages of gold won, 332, 333, 444, 484
percentages of ore treated, 333

Fault, definition of, 94
Faults (see Dykes and Faults)
Ferreira, auriferous dyke, 112
cross-cuts, cross-cutting, 296
crushing, costs, 428
development, cost, 299
dips, 12
driving, rate, 291

Ferreira, dykes, 116
explosive consumed, 362
headgear, practice, 225, 226, 232, 233, 235
reefs of, 70, 72
sample book, 313, 315
sorting, costs, 410, 412, 414, 423
tipping arrangements, 233, 235
Ferreira Deep, shaft-top, 173, 220
shaft-sinking, 163
Filling stopes (see Stulls and Stulling)
Fines (fine ore), 404, 407, 413
Flattening of dip, 12, 14, 16, 106
Florida, dips across formation, 23
reefs at, 22, 42, 88
Francke, M., 389
Frecheville, R. J., 42
Friesmantel, Otterkar, 39

Ganger, 431
Gates or dogs, 258, 260
Gates crusher, 310, 425
Gatsrand formation, the, 33
extent and dip, 33, 35
stratigraphy, 34
Gelatine, 362
Geldenhuis Estate, accounts, 440
assay plan, 317
development, cost, 299
fault and dyke, 99
headgear practice, 223, 406
hoisting costs, 269
mechanical haulage, 400
reefs, 29
sample book, 313, 315
stopes and stoping, 344, 353
transport, 402
value of ore body, 328
Geldenhuis Deep, mill samples, 331
reefs, 327
value of ore body, 328, 334
value of ore milled, 411
General charges, 431, 432
Geological Society of South Africa, 38
Geological formations, 18, 37, 38
George Goch, cross-cut, 295
extraction, 331
development, cost, 296
mill samples, 332
ore bins, 212
reefs, 75
stopes and stoping, 343
George Goch Dyke, 102
Fault, 98
Gibson, Walcot, 37, 39
Glen Deep, reefs, 79
shaft-sinking, 192
Gold, distribution, origin and occurrence, 42, 54, 62, 66, 67, 74, 91, 92, 111, 123-128
fineness, 127
Gold Estate Reef, 58
Gold Law, 1, 445
Goldmann, Sydney, 451
Government Reef, 22
Grahamstown Dyke, 99, 101
Granite formation, the, 18
Graphical method of averaging samples, 323
Great Western Reef, 57
Greening sorting belt, 422
Green's Reef, 59
Grizzlies, 240, 407, 413, 414
Ground tenure, 447
Guide frame, for bucket rope, 245
Guides, shaft-timbering, 169

Hall, W., 271
Hammond, John Hays, 388, 464
"The Genesis of the Witwatersrand Banket," 123-128
Hancock, Strangman, 385
Hand drilling, 164, 193, 288, 293, 339, 361
Hanging bolts, 168, 173-175
Hatch, F. H., 328
Hatch and Chalmers, 28, 44, 49
Headgears, Chimes West, 220, 227, 237
City and Suburban, 220, 227, 237
construction, 237, 238
dressing floors, 220
Driefontein, 220, 221
Ferreira, 225, 226, 232, 233, 235
Ferreira Deep, 220
grizzlies, 240
headgear practice, 219, 223, 226
Knights Central, 221, 224
ore bins, 223, 240
pulleys, 237
Rand Mines, 220, 222, 223
Robinson, 220
Robinson Deep, 221, 240
tipping arrangements, 236
Heidelberg district (see Nigel-Heidelberg district), 6
Heidelberg-Roodepoort Reef, 88
Henry Nourse, development cost, 296
reefs, 75
Hirnant air compressor, 368
machine drill, 378
Hoisting, 244
adjusting rope length, 263, 264
cages *versus* skips, 266
costs, 268, 270, 465
practice above surface, 219, 226
practice in deep levels, 264, 267, 268
practice underground, 210
rate of, 262, 263
safety appliances, 248, 252, 256, 257, 265
shaft signals, 266
track, 259
Hoists or winding engines, 262, 263
air hoist, 265
brakes, 262
depth indicators, 265
drums, 261
electric hoists, 265
signals, 266
sizes, 262
types, 261
Horizon of the richer reefs, 5, 9-11

Igneous and metamorphic rocks, petrological characters, 18-20, 28, 41, 48, 59, 65, 100, 102, 103, 108-110, 112, 113, 115
Illumination, cost, 391

Inclination (see also Dip), 11
Industrial Commission, 271, 361, 442, 448, 450, 451
Ingersoll-Sergeant compressor, 368
machine drills, 377, 378
Intermediate drives, 287, 357

Jack-head pump, 275
Jennings, Hennen, 442, 448
Jennings, S. J., 418
Joel Reef, 55, 63, 64
Johns, J. H., 361, 412
Jointing, timbering, 167-169, 182
Jubilee, shaft stations, 198
skip, 246
Judd, Professor J. W., 28
Jumpers, hoisting costs, 269
stopes and stoping, 357, 383
Jumpers Deep, hoists, 262, 263
development, 304, 385
machine work, 385
shaft-sinking, 154, 184, 195
Jupiter, shaft-sinking, 193

Keely and Africander Series, 62
Kimberley Series, 47, 52, 56, 57, 60, 61, 63
Klerksdorp district, 8, 10
Black Reef, 64
dip of beds, 10, 16
reefs, 57
Knights Central Deep headgear, 237
Knights Deep, shaft-sinking, 192
Knights Longitudinal Dyke, 104
Krugersdorp district, 8, 10
Black Reef, 66
dip of beds, 16, 22
reefs, 61
Kubale, G., 65

Labour, 448
coefficient of duty, 449
cost per ton milled, 485
percentage of total cost, 448
Lagging, timbering, 166, 173
Lamps, 391
Lancaster, bore-holes, 138, 140
Botha's Reef, 91
dykes, faults, 122
transport, 403
Langlaagte Estate, explosions, 362
reefs, 151
shafts, 151, 154
stopes, 345, 362
Langlaagte Royal, round shaft, 188
Langlaagte United, reefs, 88, 89
reefs and dykes, 117
Left-handed dislocations, 95
Leggett, T. H., 223, 420
Levels (see Drives)
Little Giant machine drill, 361
Living, cost of, 451
Livingstone Reef Series, 45, 52, 55, 61, 62
Loading, 198, 202, 205, 249
Loading station, 197
complete, 216
location, number, 210
practice in dealing with the ore at different levels, 210
Locomotives for transport, 401, 403
Lodges, sumps, 188, 189
Longitudinal dislocations, 94
age of, 115
dip, 106, 115
duplication by reverse faulting, 24, 47, 81, 101-105, 108
effect on the deep-level reef, upthrows by reverse faulting, 99-106, 152, 463, 466, 467
effect on the reef at surface, 108
origin of, 106, 107
peculiarities in strike, 99, 100
principal ores, 100-108
Longitudinal reverse faulting, 96, 107, 108, 463, 466
Long wall stopes, 336, 343

Machine drilling, 193, 288, 291, 299, 300, 348, 360
cost, 382, 383, 386
Machine drills, construction, 375
cost, maintenance, 382, 383
cost, working, 382, 383
duties of, 382, 383
efficiency, 384
mountings, 378, 379
types, 361, 377, 378, 385
values, 375
Machinery, cost of, 458
Main Reef, the, 45
at Johannesburg, 68
Benoni to Modderfontein, 86
correlation, Nigel Reef, 54, 68
improvement east of Johannesburg, 74
west of Johannesburg, 88
Main Reef Leader, the, 45
at Johannesburg, 70
Benoni-Van Ryn, 86
west of Johannesburg, 88
Main Reef Series, 68
correlation, Botha's Series, 61
Nigel Reef, 54
Van Ryn Series, 85
extension, 93, Fig. 29
features, 68-72, 93
in Banket (mine), 90
Glen Deep, 79
Henry Nourse, 75
Klerksdorp district, 60
Krugersdorp district, 61, 90
New Blue Sky, 84
New Comet, 81
Roodepoort District, 89
Rose Deep, 77
Van Ryn district, 50, 53, 85
Van Ryn Estate, 86
Venterskroon district, 56
Witwatersrand (Knights), 80
minor features of, 42, 48, 65, 67, 73, 74
reefs of, 45
stratigraphy, 44, 61, 85
west of Johannesburg, 88
Manager, 430
certificate as, 431, 471
examination for, 474
qualifications of, 431
Material and stores, 457

May Consolidated—
bastard reef, 110
effect of machine stoping, 360
M'Dermott, Walter, 264
Measures, square, linear, 2
Mechanical haulage, 398
Metropolitan Fault, 98, 103
Meyer and Charlton, development, cost, 296
sorting belt, 422
trial rushing, 330
Meyer and Charlton Dyke, 100
Midas Deep bore-hole, 64
Middle Reef, 45, 72
Mill samples, 331
Milling thickness of a reef, 301, 319
Mine sampling, assay plans, 316
ore valuation, 318
practice in, 305-310
record of results, 312
sample books, 313
sample grinding, 310
sample panning, 311
Mine water, amounts, 271, 272, 463
analyses, 272, 273
methods of dealing with, 273
Mining Regulations, 349, 429, 445, 470-474, 481
Minister of Mines, 445
Mjnpacht, 447
Molengraff, Professor, 28
Molyneux Reef, 21, 43
Monarch Series, 61, 62
Moore, Wilson, 28

Native labour, 453
accommodation, rations, 456
cost per ton milled, 485
supply, 456
wages, 454
New Blue Sky, reefs, 84, 85
trial crushing, 330
New Chimes, reefs, 50-52
New Comet, accounts, 438
mill samples, 331
reefs, 81, 82
New Heriot, reefs, 76, 327
New Kleinfontein, mill samples, 331
reefs, 331
sorting belt, 432
stopes, 341, 347
New Midas Estate, reef, 66
stopes, 343, 346
New Modderfontein, depth Van Ryn Reef, 153
New Primrose, bedded dyke, 100
circular shaft, 186
hoisting costs, 269
reefs, 327, 343
shaft-sinking, 164, 193
sorting belt, 422
stopes, 344, 345
New Rietfontein, reefs, 41, 327
Nigel, landing practice, 218
reef, 346
stope, 358
Nigel district (see Nigel-Heidelberg), 16
Nigel-Heidelberg district, 6, 10
dip of beds, 10, 16, 20
Nigel-Heidelberg district, reefs, 54
Nigel Reef, 54, 63, 88
Normal Fault, 95, 318
North Reef, of Main Reef Series, 45, 72
Nourse Deep, bore-hole, 139
hoisting practice, 265, 268
mine water, 272, 273
pump, 274
reefs, 76
shaft-sinking, cost, 192
value of ore milled, 334

Oceana Reef, 36, 60
Odin Reef, 57
Orange River Colony, 8
Ore bins, 203
binning before sorting, 223, 226, 227, 241
capacity, 212, 223, 241
construction, location, 203
doors, shoots, 214, 242
for incline shafts, 205
for vertical shafts, 203
mouths of, 207, 210
on headgears, 241
Ore-dressing, 404, 436
Ore shoots, pay shoots, 42, 65, 67, 125
Ore valuation, by mine sampling, 305
adjustments, 321
averaging different layers, 318
averaging along a length, 319
averaging over an area, 325
estimation and recovery, 328, 334
graphic averaging, 323
limitation, incomplete exposure, 326
limitation, patchiness, 327
relative value of a reef, 318
by reduction results, 332
by sampling the broken ore on surface, 330
by trial crushings, 329
Outcrop mines, 2
economics, 459, 460
mine water, 271
shafts, 150, 153
value, ore body, 459
value, ore milled, 462
Outcrop and deep levels, 2
comparison, ore body, 461
recovery value, 462
working costs, 464, 465
Overhand stopes and stoping, 335, 339, 340, 350, 357, 358
Overlap faults and faulting, 79, 95, 96
Overseers, 431
certificates, 473
examination, 476
Overwinding, 257

Panning, 311
Payable ore body, 459
Pay shoots (see Ore shoots)
Pebbles (see Banket)
Pettit, A. E., 205
Phillips, Lionel, 302, 438
Pillars, 346
economy of, 340, 347, 482
proportion of ore as, 301
robbing, 348
service of, 348

Pitchford, J. B., 186, 256
Pitwork, Cornish, 279, 280-283
Platt, definition, 197
Plunger poles, 282
Prices of stores and materials, 457
Princess Estate, development cost, 300
stopes, 342, 362
Profit per ton, 442, 444, 445
Prospecting, 129
Pump columns, 283
chambers, 188, 274
rods, 279, 281
stations, 274
Pumping, cost, 284, 465
lifts, 273, 274
power required, 283
Pumps, air pumps, 274
Cornish, 275
electric, 284
force, 284
jackhead, 275
steam, 283
three-throw, 275, 284
Pyrites, iron pyrites, 42, 48, 59, 62, 67, 71

Quartz, auriferous, 43, 74, 127
veins, 43, 54, 74, 109, 110, 112, 127
Quartzite, auriferous, 51, 68, 69, 309, 408
nature, 25, 33
Quartzite-Shale group, 20
banket beds, 21
component beds, 20
dip, 20, 22
extent, 20-22, 34
stratigraphy, 23

Rails, 394
Raise, 292
Rand Drill Company compressor, 368
drills, 377
Rand Mines, Limited, daily wage, 450
development, 285
headgears, 220
values of claims, 468
Rand Victoria bore-hole, 139, 142
Recovery values of the goldfields, 442, 444
Reduction, range of the term, 4
Reeds elevator, 403
Reef cores, 140
value of the assays, 139
Reef claims (see Claims)
Reef values, average, 318
relative, 318
Reefs (see Banket Beds)
Residues, value of, 410, 478, 483
Responsibility in mines, 470
Reverse faults and faulting, 79, 95, 96, 318
(see also Longitudinal Dislocations)
Rietbult reefs, 23, 43, 88
Rietfontein A reefs, 41
Rietkuil Syncline, 36
reefs, 59
Right-handed dislocations, 95
Roberts, C. T., 369
Robertson, A. R., 156
Robinson, development costs, 297
headgear practice, 220, 226
hoisting, costs, practice, 211, 269
Robinson, shafts, 150, 151, 183, 192
stations, ore bins, 200
tramming, 287
winzing, 293
Robinson Deep, headgear, 221, 237, 240
hoisting, 265
mine water, 271, 272
ore bin stations, 203, 204, 205
reefs, 72, 73
sample book, 314
shafts, 162, 166, 192
temperatures, 388
value of ore milled, 334
Robinson Dyke, 100
Roodepoort Central Deep shaft, 154, 177, 189, 195
Ropes, 246, 269, 399, 480
Rose Deep, bore-hole, 139
mine water, 272
reefs, 77
value of ore milled, 334

Safety appliances, 231, 233, 248, 252, 256, 257, 265
Salaries, 449
Salisbury Dyke, 113, 116, 119
Sample bags, 309
books, 312
grinders, 310, 311
Samplers, 305
Sampling (see Mine Sampling), 330
Sampling reef cores, 140
Sawyer, A. R., 39
Schenck, Dr., 39
Schweder, K., 368, 384
Screening grizzlies, 407
Secondary steepening in depth, 13, 14, 15, 106
Sets of timbering, 165
Seymour, L. I., 382, 383, 448, 450, 458
Shaft pillars, 348
Shaft signals, 266
Shaft-sinking, 155
at the Angelo Deep, 162, 196
Durban-Roodepoort Deep, 157, 194
Ferreira Deep, 163
New Primrose, 164
Robinson Deep, 162
Simmer East, 161
Vogelstruis Consolidated Deep, 156, 195
comparison, hand, machine, 194
cost, 189, 190, 192, 193
methods of drilling, 155
on the incline, 164
rate, 191, 193, 195
ventilation, 387
Shaft timbering, 165
angle connection, 184-187
at Angelo Deep, 162
at Roodepoort Central Deep, 176
bearers, 173, 175
cap pieces, 178
collar set, 173, 183
cost, 189, 190
distance pieces, 178, 183
dividers, 165, 168, 169
end plates, 165, 166

Shaft timbering, fixing and supporting, 173
hanging bolts, 168, 173, 175, 183
hitches, 175
guides, 169, 183
incline, 178
jointing, 166-169, 182
lagging, 166, 173, 183
procedure, 176
sets, 165
sole pieces, 178
studdles, 171
wall plates, 165, 166
wedging and blocking, 176
Shaft tops, landing of skip and cage, 236
Shafts, costs, 189, 194-196
circular, 186
location, 150, 151
number, 469
sizes, 152, 155, 191
Shales, occurrence and nature, 22, 25, 34, 48, 49, 53, 66, 113
Shift, working day, 157, 291, 363, 364, 445, 449
Simmer and Jack, development cost, 298
dips, 13
hoisting costs, 269
reefs, 327
transport, 397, 402
value, ore body, 328
Simmer and Jack East, bore-hole, 105, 139
shafts, 161, 186, 187, 193, 195
Simmer and Jack West, shafts, 154, 186, 193
Simmer Dyke, 103
Skips, attachment of rope, 251
capacities, 248, 250
construction, 248-251, 256
framed skips, 247
self-dumping skips, 246
wheeled skips, 249
Slate Leader, 108, 110, 216
Slate parting, bedding plane, 70, 72, 78, 81, 83, 108, 110
Sleepers for track, 395
Slugger machine drill, 377
Smith, Hamilton, 388
Sorting, at the Crown Reef, 417
,, Ferreira, 410, 414
,, Treasury, 420
advantages, effect, 409, 412, 478, 482, 483
costs, 412, 416, 418, 419
methods, 412, 423
shaking tables, 422, 424
sorting belts, 420
sorting floors, 228, 407, 414, 424
sorting tables, 417, 424
underground sorting, 350, 409, 412
working test, 478
South African Association of Engineers and Architects, 28, 143, 368, 369, 382
South Reef, the, 45
at Johannesburg, 71
Johannesburg to Treasury, 77
Johannesburg to Van Ryn, 86
deterioration going east, 50, 76
in the Botha's Series, 92
in the New Comet, 82
west of Johannesburg, 88
Southern Heidelberg Syncline, 5
axis, 17
dip, 18
extent, 17
geological formations, 35
reefs, 63
South Rand Dyke, 102
Specific gravity of banket, 66, 71
Spes Bona, bedded dyke, 108, 109
shaft, 153
Springbok Reef, 57
Stable Reef, 42
State Mining Engineer, 44, 65, 389, 445, 449
Station, 197
Stations, 197
construction, location, 197
for incline shafts, 198
for vertical shafts, 198
Statistics of the output, 44, 445, 446
persons employed, 446
Steepening in depth (see Dip)
Stope, 335
boxes, 354
drives, 287, 341, 347
pillars, 342, 347
plans, 365
timbering, 354
Stopes, kinds, 335
sizes, 72, 345, 358, 359, 363
Stoping, cost, 360
explosive consumption, 361
practice, 344, 350, 359
shovelling, 357, 363
stulls and stulling, 349, 352
Stoping drills, 378, 383
Stoping thickness, 319
Stores and material, 457
percentages of expenditure, 433
prices, 457
Strike faults, 94
Striped Pebble Reef, 55
Struben, 36, 39
Studdles, 166, 171
Stulls, stulling, 338, 349, 352, 482
Sub-Nigel, Ltd., reefs, 55
Sullivan diamond drill, 133
Sumps, 188, 274
Surface foreman, 431
Surveying bore-holes, 143
Sweep rods, 278, 279

Tandem crushing, 413, 427
T bob, 279
Temperature underground, 142, 388, 463
Theories of origin of banket beds, 123
gold in banket, 124, 125, 126
Ton, weight, 3
Tonnages per claim, 484
Topography, 129
Track, 394
Tramming, 394
cost, 402
Transport, 394
by animal power, 401, 402
by mechanical haulage, 398, 402
by traction electric power, 402
by traction steam power, 402
costs, 401, 402

Transverse dislocations, 94
dip, 100
Krugersdorp district, 97, 122
Roodepoort district, 98
Van Ryn district, 96, 121
Treasury, ore-dressing plant, 420
reefs, 77
Trenching, 130
Trial crushings, 329
Tribute work, 364, 453
Tributor, 429
Trucks, 395
capacity, 397, 402
construction, 396
Tumbler, 233, 234

Underground temperature, 389, 463
Underhand stopes and stoping, 335-337, 350, 357, 358
Upthrows, 101-105, 107, 152, 466, 467

Valuation (see Ore Valuation)
Van Ryn-Chimes district, 6, 10
dip of beds, 10, 15
reefs, 50
Van Ryn Estate, dykes and faults, 121
reefs, 86
Van Ryn West, dykes and faults, 121
reefs, 52
Venterskroon district, 8, 10
dip of beds, 10, 16, 20
reefs, 56
Ventilation, 387
Village Main Reef fault, 105
pump, 283
shaft, 155
Vogelstruis Consolidated Deep, shafts, 156, 172, 176, 192, 194, 195

Wages, native, 454
white, 157, 449-451
Walling waste for stulls, 352
Wall plates, timbering, 165, 166, 178
Waste, headgear practice, 219, 220, 229, 238, 241
necessity of mining, 407
proportion mined, 408
value, sorted out, 409, 410, 478, 483
walling, 352
Water (see Mine Water)
Weight and specific gravity of banket, 3, 66, 71, 301
Wemmer, dips, 70
sorting plant, 424
transport, 401
value of the reefs, 69-72
West Rand Mines, North Leader, 91
stopes, 345
West Roodepoort Deep, shafts, 196
reefs, 89
Wheels and axles, 397
White, H. B., 189, 300
White, Franklin, 3, 71
White labour, 449
accommodation, 451
cost per ton milled, 485
statistics, 450, 451
wages, 447, 451
Whiting's hoist, 264
Wilson, N., 223, 420
Winzes, 294
in the Crown Reef, 293
costs, 294
distances apart, 292
driving, 292, 293
Witpoortje Break, 9, 90, 97, 98
Witwatersrand (Knights), accounts, 437
reefs, 80
Witwatersrand Beds, 23
age, 36
outcrop thicknesses, 24, 34, 35
reefs, 25
stratigraphy, 23
Witwatersrand Deep, shafts, 154
Witwatersrand Goldfields, 1
hills, 1
Syncline, 5
axes, 9
districts, 6
extent, 9, 10
geological formations, 18, 34
inclination of beds, 11
Worcester, bedded dyke, 110, 111
stations, 202
Worcester Hope Reef, 59
Working costs, 437, 439, 441, 442, 464, 465, 482, 485
day (see Shift)
Wright, B. H., 229

THE END

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